MINERALS

ECONOMIC & REGIONAL DEVELOPMENT AGREEMENT

Final Report
TASK 1A - VBM Design Guidelines
Chapter 6

by

INCO Limited, Thompson Division
Thompson, Manitoba

December 1989

Canada

Manitoba

MG-13097A

CANADA - MANITOBA
MINERAL DEVELOPMENT AGREEMENT
RESEARCH & DEVELOPMENT IN MANITOBA MINES

Work on the project was conducted under the auspices of Canada Centre for Mineral and Energy Technology, Energy, Mines and Resources Canada under the Canada/Manitoba Minieral Development Agreement.

DSS File No. 14SQ.23440-4-9147-3
DSS Contract Serial No. OSQ84.00403
CANMET Project No. 4-9147-3
CANMET Scientific Authority: Y.S. Yu

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MINERAL DEVELOPMENT AGREEMENT
RESEARCH & DEVELOPMENT IN MANITOBA MINES

VERTICAL BLOCK MINING (VBM) DESIGN GUIDELINES

Chapter 6

VBM Stoping Procedures

This chapter has been prepared as part of the

TASK 1
To Develop Vertical Block Mining Design Guidelines
For A Steeply Dipping Orebody;
And To develop Scaling Techniques
of the
Canada/Manitoba Mineral Development Agreement

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DSS File No.: 14SQ.23440-4-9147-3
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MINERAL DEVELOPMENT AGREEMENT

RESEARCH & DEVELOPMENT IN MANITOBA MINES

VERTICAL BLOCK MINING (VBM) DESIGN GUIDELINES

The Vertical Block Mining (VBM) Design Guidelines consists seven chapters, published separately. The seven chapters are:

Summary

1. **Cut and Fill Mining**
2. **Data Collection Systems**
3. **VBM Stope Design**
4. **Drilling and Blasting**
5. **Support Requirements**
6. **VBM Stoping Procedures**

Reference to this chapter should be quoted as follows: "**Vertical Block Mining Design Guidelines Chapter 6 - VBM Stoping Procedures**"; Canada/Manitoba Mineral Development Sector B - Mining; DSS File No. 14SQ.23440-4-9147-3; CANMET Project No. 4-9147-3; December, 1989

CHAPTER 6

VBM STOPING PROCEDURES

November, 1989

852-1605

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1.0 INTRODUCTION

This section of the main task describes stoping procedures for vertical block mining (VBM) as they are practiced at Inco's Thompson Mine in Manitoba. A description of stoping procedures at other similar operations is also provided to illustrate practical mining solutions which have evolved in somewhat different geologic environments. The goal of this work is to provide workable solutions to problems which will likely be encountered during the implementation of VBM stoping in a steeply dipping orebody in any hard rock mining environment.

2.0 OBJECTIVES

The Thompson Mine has been in operation since 1961. From that time, until approximately 1980, the mine was primarily a cut-and-fill operation. In the early 80's, vertical block mining (VBM) was introduced in order to:

1) enhance safety by removing miners from the stopes (non-entry) mining;

2) to increase productivity and reduce per ton operating costs.

The conversion has progressed to a stage whereby VBM contributes upwards of 80% of the present underground mine production.

The method has generally been successful. Productivity has increased, together with a greatly improved safety performance. On the other hand, however, though the method has realized a lower per ton operating cost, dilution has increased, resulting in reduced grades.

Like all mining, VBM at Thompson has evolved over time by a trial and error process. At this particular point, the method has proven to work well, but it is recognized that further modification and optimization are required. Thus, it is anticipated that the procedures outlined here will change.

A number of constraints exist at Thompson, which influence the application of the method. Thus, great care must be exercised if the Thompson procedures are to be adopted elsewhere. The constraints include:

1) narrow tabular style orebody that pinches and swells;

2) mature mining; therefore existing services sized around initial mining method, limited mobility for trackless equipment; significant portion of ore tonnage derived from secondary and tertiary mining areas;

3) extended nature of mining; that is many workplaces separated both laterally and vertically with limited access between each for trackless equipment;

4) conversion process.

Given the above constraints, the objective of this study is to outline the VBM procedures used at Thompson, the achievements obtained and compare these with other mines using the VBM method. This will allow areas for further modification and optimization to be identified.

3.0 VBM STOPING AT THOMPSON MINE

3.1 General

VBM refers to the concept of mining narrow orebodies with small vertical blocks, then retreating longitudinally following placement of cemented hydraulic backfill (figure 6-1). It does not specify methods of drilling and blasting. Currently, most ore in VBM stopes at Thompson Mine is broken by vertical crater retreat (VCR) blasting since it offers significant advantages over other methods. The following stoping procedures therefore refer to those practiced with VCR blasting.

Production at Thompson is currently at a rate of 5,442 tonnes/day (6,000 tons/day). Figure 6-2 illustrates a longitudinal section of the mine. The nickel ore is a massive sulphide located in a fairly competent schist zone which is conformable with the local geologic units. The ore pinches and swells to a large degree, ranging in dip and thickness over the entire 5.6 km (3.5 mile) strike length (Sarin, 1981). Orebody characteristics are summarized as follows:

Orebody width:	1.5 - 30.5 m, 9.2 m (5 - 100 ft., 30 ft.) average
Orebody dip:	45 - 90^{O}, 69^{O} average
Average mining grade:	2.7% nickel
Mining depth:	183 - 1220 m (600 - 4,000 ft.) below surface
Unconfined compressive strength of ore:	70 MPa
Hanging wall rock types:	73% schist
	15% quartzite
	4% peridotite
	8% other

Footwall rock types: 46% schist

34% quartzite

16% peridotite

4% other

As can be seen above, schist is the dominant hangingwall and footwall rock type. Thickness of schist units is variable, with typical values in the order of 6.1 m (20 ft.). Rock mechanics of the mine is described in more detail elsewhere (SRK, 1984).

The breakdown of mining methods in use at the Thompson Mine during 1989 are as follows:

Method	Workplaces	Tonnage
VBM	20	82%
Cut-and-Fill	4	14%
Pillar Recovery	5	4%

3.2 Development

VBM mining requires the development of an overcut drill drive and an undercut mucking level for longitudinal mining. Drawpoints and haulage drift are required for transverse mining. Raise and slot development are eliminated because the undercut provides the free face for blasting.

Minimum drill drift dimensions are 3.7 m (12 ft.) wide x 3.7 m (12 ft.) high to accommodate ITH drills. Wider overcuts are developed where ground conditions permit in order to obtain a greater degree of parallel hole drilling. Single drill drifts are generally driven along the hangingwall contact. Where orebody width is excessive, two roughly parallel drill drifts can be driven, with one along each contact (figure 6-3).

When the overcut drift has been completed, a 14.3 mm (9/16 inch) cable is anchored at either end of the block, generally to the back. This cable is later used to anchor safety lanyards during loading and blasting.

The extraction drift is generally driven in footwall rock, up to 30.5 m (100 ft.) from the orebody. Previously, drifts were much closer to the stoping areas. Major problems with stability were experienced. Thus, to avoid costly rehabilitation, the decision was made to locate the drifts at a greater distance into the footwall. Drawpoints are driven off the extraction drift into each block. Where hangingwall rock is more competent, the extraction drift is sometimes driven in the hangingwall.

Large VBM stoping areas are serviced by a ramp access to facilitate movement of mobile equipment and supplies. Ramps are driven at a maximum grade of 20%, and connect all sublevels.

VBM stope dimensions are determined by the prevailing ground conditions, stress levels and orebody geometry. The minimum stoping width is about 3.7 m (12 ft.) Stopes with widths up to 12.2 m (40 ft.) are mined longitudinally. Wider stopes are mined transversely. Apart from these general guidelines, there are numerous variations to the extraction method imposed by structural and geometrical considerations.

Inco has collected data on stope dimensions mined at Thompson since the introduction of VBM in 1979. This data is presented as a series of histograms in Appendix 6-A, and is summarized as follows:

Stope width:	1.5 - 30.5 m, 9.2 m (5 - 100 ft., 30 ft.) ave.
Stope strike length:	6.1 - 41.2 m, 18.3 m (20 - 135 ft., 60 ft.) ave.
Stope height:	15.3 - 61 m., 40 m (50 - 200 ft., 130 ft.) ave.
Stope tonnage:	4,535 - 63,490 tonnes, 18,140 tonnes (5,000 - 70,000 tons, 20,000 tons) ave.
Drawpoint to orepass haul distance:	30.5 - 336 m., 104 m (100 - 1,100 ft., 340 ft.) ave.

Standard drift dimensions for drawpoints, extraction drifts, ramps and ore pass accesses vary depending on the size of LHD being used. Minimum drift sizes are quoted as follows, and do not include any allowance for ventilation duct:

2 cu. yd. (1.5 m^3) LHD:	3.1 m (10 ft.) wide x 3.2 m (10.5 ft.) high
3.5 cu. yd. (2.7 m^3) LHD:	3.7 m x 3.2 m, 18.3 m (12 ft. wide x 10.5 ft.) high
5 - 6 cu. yd. (3.8 - 4.6 m^3) LHD:	4.0 m x 3.2 m (13 ft. wide x 10.5 ft.) high

Drift heights can be reduced to 2.7 m (9 ft.) where they are not used for ITH drilling, tramming or truck usage. A listing of mobile equipment used at Thompson is enclosed in Appendix 6-B.

Holes for drift development rounds are drilled from 2.4 - 3.7 m (8 to 12 ft.) deep at 44 mm (1.75 inches) diameter. ANFO is blown into the holes with compressed air, and Powertrim cartridges are loaded in the back holes (Golder, 1987). Initiation is via LP Nonel delay caps. Permanent drifts are sometimes mined with an arched back for ground control purposes. Lifters are drilled to break 0.31 m (1 foot) below grade so that ballast can be added to produce a good roadbed at grade. Auxiliary ventilation tubing is maintained within 15 m (50 ft.) of the face (Inco Training Manual, 1989). About half of the development headings are drilled with electric/hydraulic jumbos, while the other half are drilled with hand-held jack-leg drills.

All drifts in ore are bolted and screened. The normal rockbolt type is a double threaded mechanical bolt, 2.4 m (8 ft.) long. Headings are kept bolted to within 0.92 m (3 ft.) of the face. Where ground conditions dictate,

5.0 m (16 ft.) extension bolts are sometimes used. In weak ground, split sets are often used. In extreme situations, cable slings, consisting of a grouted Split Set and cable combination, are installed.

In VBM stopes, extra ground support is installed as is required. Cable bolts are installed in drawpoints and overcut drill drifts. Cable bolt lengths vary from 6.1 - 12.2 m (20 - 40 ft.) They are well suited for ground control in VBM stopes, and are used in soft ground (along with grouted rebar) where it is difficult to get expansion shell anchors to hold securely. They are especially important in large overcut drill drifts, where parallel hole drilling is to be carried out in wide ore bodies (figure 6-4).

Drifts in waste rock are usually screened. Standard rockbolts used are 1.8 m (6 ft.) and 2.4 m (8 ft.) mechanical bolts. Permanent headings in waste are protected with shotcrete if ground conditions warrant.

3.3 Production Drilling

All production holes are drilled down from the overcut drill drift, using in-the-hole (ITH) hammer drills. Hole sizes drilled are 114 mm (4.5 in.) and 165 mm (6.5 in.) diameter. Ideally holes are drilled parallel to the hangingwall and footwall contacts and break through into the undercut drift (figure 6-5). In practise, holes are drilled in an inverted fan pattern with many perimeter holes being non-breakthrough. In order to accommodate irregular ore/waste contacts, drilling angles range from vertical to about 55^{o} from horizontal.

Engineering layouts are supplied for all production drilling. The planned maximum hole burden and spacing for VCR blasting is:

- for 165 mm (6.5 in.) holes - 2.4 m (8 ft.) spacing, 3.1 m (10 ft.) burden;
- for 114 mm (4.5 in.) holes - 1.5 m (5 ft.) spacing, 2.1 m (7 ft.) burden.

Drilling accuracy is critical to successful blasting and the control of dilution. To achieve a high degree of accuracy, it is recommended that drill alignment be set prior to commencement of drilling, rechecked after the hole has just been collared and rechecked again when the first 0.60 m (2 ft.) has been drilled. Drillhole deviation is checked by surveying breakthrough locations and plotting in plan. A maximum deviation of 5.3% was recorded for 24 m (80 ft.) long 114 mm (4.5 in.) diameter holes in one example of mining in a rib pillar (Stewart, 1988). In another case, a maximum deviation of 5% was recorded for 43 m (140 ft.) long 165 mm (6.5 in.) diameter holes (Golder, 1987).

Casing pipes are installed at hole collars to keep drill cuttings out of the hole. Compressed air from the mine is boosted with portable compressors before being supplied to the ITH hammer.

Statistics for production drilling at the Thompson Mine are summarized as follows:

Hole Size (inches)	Tonnes (Tons Broken per foot drilled)	(Feet Drilled per 8 hr. shift)
114 mm (4.5 in.)	1.4 - 4.5, 2.4 (1.5 - 5.0, 2.6) ave	24 - 52 m, 35 m (80 - 170 ft., 115 ft.) ave
165 mm (6.5 in.)	1.8 - 6.8, 4.1 (2.0 - 7.5, 4.5) ave	15 - 49 m, 32 m (50 - 160 ft., 106 ft.) ave

3.4 Loading and Blasting

Each crater blast is designed by Engineering prior to each blast. This design includes sequence, charge weight, stand-off and selection of blast holes.

Prior to loading of charges, all holes are inspected to identify any blockages or closures. If breakthrough holes are plugged, they can usually be cleaned out by dropping a heavy brass pin or bar down the hole, attached to a rope. More stubborn blockages are freed with an air hose. If these measures are unsuccessful, an ITH drill is used with a cleaning spoon on the end of the drill column.

If a blocked hole has been previously blasted, it is cleaned out via remote control drilling. Normal drilling can take place to a distance of 6.0 m (20 ft.) from the previous charge location. Within 6.0 m (20 ft.) the drill must be operated by remote control with the operator at least 4.6 m (15 feet) from the hole collar.

If all holes are open, they are measured to determine depth to the stope back and to the top of broken ore (Figure 6-6). Depths are recorded for blast design and to determine the void space present between muckpile and stope back. The recommended void space is 1.5 times the blast volume, or 2.7 m (9 ft.) for a 1.8 m (6 ft.) thick slice of ore (Inco Training Manual, 1989). Experience has shown that this provides sufficient expansion room for muck that will be broken in the next blast. Hole measurements are also used to determine if the previous blast broke as intended.

The loading configuration for VCR blasting is shown in Figure 6-7. Holes are blocked by lowering a wooden plug attached to 8.0 mm (5/16 inch) polypropylene rope to the proper depth, then securing the rope in the overcut drift. Plug sizes are 76 x 76 x 305 mm (3 x 3 x 12 in.) for 114 mm (4.5 in.) holes and 127 mm (5 in.) diameter x 305 mm (12 in.) for 165 mm (6.5 in.) holes. A minimum of 0.31 m (1 ft.) of stemming is then shovelled into the hole. Stemming materials used include sand, drill cuttings or slag.

The distance from the centre of the explosive charge to the hole bottom is called the "stand-off" distance. This distance is critical for optimal blasting, and varies with the type of ground, hole size, hole pattern and explosive type. The normal stand-off for 25 kg (55 lb.) Aquamex charges in 165 mm (6.5 in.) holes is 1.8 m

(6 ft.), and in 114 mm (4.5 in.) holes is 1.2 m (4 ft.). This distance can be varied. For example, in 1403-301 block, the standoff distance used in 114 mm (4.5 in.) holes ranged from 0.9 - 1.4 m (3.0 - 4.7 ft.).

Loading procedures involve slitting open a packaged emulsion/slurry explosive to provide a high degree of coupling to the rock. Half the charge is dropped down the hole, the primer is placed on top of the charge, and the remainder of the charge is dropped into place. Stemming is then shovelled on top of the charge. A similar procedure is adopted for ANFO. Design criteria for charging at Thompson are summarized as follows (Inco Training Manual, 1989):

Design Parameter	114 mm (4.5 in.) Hole	165 mm (6.5 in.) Hole
Charge weight (Aquamex or ANFO)	8.0 kg (18 lb.)	25 kg (55 lb.)
Stand-off distance	1.2 m (4.0 ft.)	1.8 m (6.0 ft.)
Booster type with down-the-hole delay	Mini-slider primer	Regular slider primer
Stemming length above charge	3.1 m (10 ft.)	3.1 m (10 ft.)

The stemming length evolved over time. Initially, the distance was about 2.5 m (5 ft.), but problems with cratering at the hole collars necessitated an increase to the current 3.1 m (10 ft.) lengths.

Blasts are initiated by either a down-the-hole delay system or collar delays in conjunction with detonating cord downlines. Down-the-hole delay utilizes the CIL slider primer system, which consists of a 450 gm cast booster, ms Nonel delay detonator and 5.3 gm/m E-cord. Collar delays utilize detonating cord (varying from 5.3 gm/m E-cord to 10.6 gm/m reinforced Primacord) with a 230 gm Pento-mex booster. Holes are tied in with E-cord. All blasts are initiated electrically using an electric blasting cap attached to the E-cord array.

The first VCR blast in a stope utilizes the undercut drift for expansion room. As the blasting progresses upwards, sufficient swell is mucked to maintain expansion room for subsequent blasts. Non-breakthrough holes are slashed (using crater charges) into the void when the breakthrough holes have been VCR blasted to a sufficient height (Figure 6-8). The blasting and mucking cycle is repeated until 7.6 (25 ft.) of ore is left below the overcut drill drift. This ground is taken out in one blast using decked charges separated by stemming.

The loading procedure for a final breakthrough blast is shown in Figure 6-9. Charges in adjacent holes are staggered to attain a more even charge distribution. Multiple charges in one blasthole are separated with stemming, as shown.

The blast firer is responsible for ensuring that all affected areas are cleared and guarded before the blast is fired. A blasting letter is issued by the Engineering Department for all blasts in excess of 900 kg (2,000 lb.) of explosive, detailing ventilation arrangements and any other precautions to be taken. Generally, all major blasts are conducted at midnight on a Friday to permit exhausting of fumes over the weekend.

Inco has carried out an analysis of powder factors used in VBM stoping at Thompson Mine. Histograms are presented in Appendix 6-A, and results are summarized as follows:

114 mm (4.5 in.) Holes:	PF = 0.25 - 1.15 kg/t (0.5 - 2.3 lb./ton), 0.52 kg/tonne (1.03 lb./ton) ave.
165 mm (6.5 in.) Holes:	PF = 0.15 - 1.15 kg/t (0.3 - 2.3 lb./ton), 0.51 kg/tonne (1.02 lb./ton) ave.

All secondary blasting of oversize is initiated electrically. These blasts are limited to lunch time or end of shift. A supervisor must be present if an employee could be exposed beyond a drawpoint brow while placing a charge. The blast must be properly guarded, and any oversize which can't be inspected on all sides for a potential missed hole must be drilled by remote control.

3.5 Mucking and Production Control

Mucking at Thompson is carried out with 1.5, 2.7, 3.8 and 4.6 m^3 (2, 3.5, 5 and 6 cubic yard) LHD's. Ore is mucked through drawpoints (transverse retreat) or within the stope (longitudinal retreat) as shown in Figure 6-5. Ideally, during blasting cycles, the stope should be kept full of broken ore to provide wall support, allowing LHD's to muck ore without entering the stope. However, this ideal is rarely achieved due to production constraints. The end result can be excessive dilution from the hangingwall and adjacent backfilled blocks, and the need for a large percentage of the substantially slower remote control mucking.

Warning lines are painted on the drift walls to mark the limit of LHD operator exposure. Whenever an opening occurs at a drawpoint brow, such as during final clean up of the stope, LHD's must be operated by remote control. During remote operation, operators must stay at least 7.6 m (25 ft.) back from the brow.

Statistics for mucking at Thompson are as follows:

Tons mucked per 8 hr. shift:	90 - 1179 t/s, 426 (100 - 1300 t/s, 470 tons/shift) ave.
Drawpoint to orepass haulage distance:	90 - 998 m, 308 m (100 - 1100 ft., 340 ft.) ave.

There are distinct differences between methods of grade and production control for cut-and-fill and VBM mining. For example, VBM places total reliance on diamond drilling information, whereas cut-and-fill places only partial reliance on diamond drilling due to its ability to follow unexpected orebody geometries. Methods of production control for VBM mining are summarized as follows, with those for cut-and-fill included for comparison:

Activity	VBM	Cut-and-Fill
Diamond drilling	Total reliance	Partial reliance
Orebody outlines	By interpretation	From cut mapping
Geological mapping	Sills mapped, minimum of 24 m (80 ft.) intervals	Backs mapped, 3.1 m (10 ft.) intervals
Muckpile monitoring	At drawpoint	In stope
Dilution estimation	By observation only	Survey pick-up in stope
Sand dilution	Incidental	Pre-determined
Production rates (for similar size stopes)	426 (470 tons/day)	90 (100 tons/day)

3.6 Backfilling

When mucking of ore has been completed, all VBM stopes are backfilled with cemented tailings. Backfilling forms an integral part of the mining operation by providing support to stope walls when the broken ore has been removed. This maintains integrity of hangingwall and footwall rocks, permitting extraction of adjacent VBM blocks. Tailings used for backfill (about 30% of the total mill tailings) are cycloned to the consistency of sand, and cement is added at a ratio of 25:1 sand: cement.

Backfilling procedures and barricade construction vary significantly depending on the proximity of previously backfilled blocks. Appendix 6-C includes several figures illustrating standard practice for the various situations encountered at the Thompson Mine. Details of pressure gauge, drain tower and barricade construction are also enclosed (Inco, 1988).

A typical stope preparation for backfilling is illustrated in Figure 6-10. Details of barricade construction are shown in Figure 6-11. The critical aspect of backfill control is to prevent buildup of hydraulic pressure at the drawpoint barricade(s). This is accomplished by limiting the amount of water in the VBM block at any one time and by providing drainage facilities. The rate of backfill pouring is regulated in accordance with the drainage to prevent accumulation of excess water in the stope. Water inflow and outflow are measured daily to balance water and determine the frequency of pouring.

Barricades are designed to hold backfill and allow water to drain through, thereby inhibiting build up of hydraulic pressure. Two drain towers immediately behind the barricade also help dissipate water pressures. A zero to 104 kPa (15 psi) pressure gauge is installed to monitor pressure behind the barricade. During pours it is monitored every hour. Inco uses the following criteria during backfill pours (Mann, 1982):

1) Pouring not started until water pressure below 14 kPa (2 psi);

2) Pouring stopped if water pressure reaches 28 kPa (4 psi).

Backfill is poured into the stope through 102 mm (4 in.) diameter vitaulic pipe.

The first sandfill pour is done up to the drawpoint brow, at which time all barricade leaks are sealed and the drainage system inspected. This first pour uses no cemented fill. The second and third pours bring the fill elevation to 6.1 m (20 ft.) above the top of the barricade. These pours are allowed to cure for about 4 days before filling is resumed. The remaining pours are limited to a maximum height of 7.6 m (25 ft.) of slurry at any one time. The minimum pour density required is 60% solids by weight during all backfilling (Mann, 1982). Pours cannot be started if surface water is present on the fill.

A gob fence is constructed in the overcut drill drift as shown in Figure 6-10. The overcut portion of the stope may be filled with straight sand up to the back. The backfill in the completed block must be allowed to cure for 28 days before blasting is started in the adjacent ore block.

3.7 Pillar Recovery

VBM stoping has proven to be very adaptable to the recovery of pillars at Thompson, including both rib and crown pillars. It has yielded improved safety, increased productivity and lower unit costs, than cut-and-fill techniques.

Pillar recovery at Thompson was initially planned using cut-and-fill stoping. Although it provides greater flexibility than VBM for recovery of irregular orebody geometries, it is slow and labor-intensive, especially in the poor ground conditions commonly found in pillars. It normally requires extensive use of timber during the mining cycle to protect personnel in the workplace. VBM offers a significant advantage by removing personnel from repeated exposure to these conditions, thereby improving safety and productivity. An economic analysis is currently carried out prior to all pillar recovery to determine the optimal mining method in each case.

For VBM stoping in pillars, ramp access is developed to provide continuous trackless access to the levels. In a recent example, a ramp dimension of 3.7 m (12 ft.) wide by 2.7 m (9 ft.) high was driven at a maximum grade of 30% (Stewart, 1988).

Crown pillar mining is normally carried out under a log mat and cemented fill. The log stringers are supported by 152 mm (6 inch) wide nylon straps which were bolted to the wall after the ore above was mined out. The technique has been in use at Thompson since 1982, and involves construction of a log mat with laminated stringers on 1.8 m (6 ft.) centres. Cemented fill is placed on top of the mat. When the ore below is removed, wooden posts are not required to support the laminated stringers because of the nylon strap support. The unsupported span of the log mat is not allowed to exceed 3.7 m (12 ft.).

Drilling and blasting statistics for examples of pillar mining at Thompson are quoted as follows (Stewart, 1988):

Hole size:	114 mm (4.5 inch) diameter
Drill type:	GO-60 ITH drill

Hole spacing:	1.5 m (5 ft.) at breakthrough
Hole deviation:	5.3% maximum over 2.3 m (76 ft.) length
Blasting agent:	Aquamex
Initiation:	Pentomex booster, Primacord, Nonel caps in the hole. Truck cord is B-line with electric blasting cap
Powder factor:	0.43 kg/tonne (0.85 lb./ton)
Mucking:	3.2 m (3.5 yd.) scooptrams

Rib pillars are normally recovered with cemented backfill on 2 or 3 sides of the stope. This backfill has exhibited excellent strength, with wall slough of 0.6 m (2 ft.) or less reported in one example. However, backfill has been a source of dilution in some stopes. This problem is normally easy to see but difficult or impossible to quantify.

Crown pillar recovery has been carried out by drilling uppers from the extraction level. A slot raise is developed at one end of the stope, then upholes are loaded and blasted into the slot. All activities, including drilling, blasting and mucking, occur on the extraction level. The upper level is not required. Mining commonly occurs under a previously placed log and strap mat and retreats longitudinally, thereby obtaining complete recovery of the ore. If excessive wall failure begins to occur, the open stope is backfilled and a new slot raise driven. Mining can then continue using the same technique.

3.8 Overall Performance of VBM

The increasing use of VBM over cut-and-fill stoping at Thompson is proof of its many advantages. Although cut-and-fill provides greater flexibility, VBM provides significantly higher productivities and lower cost per ton. The economic advantages of VBM are considerable even when the extra cost for development (but not dilution) is taken into account. Experience at Thompson has shown that the direct mining cost per ton, including all required development, is 20% less for VBM than for cut-and-fill (Ludwig, 1985). The advantage in productivity is more dramatic. Typical values in pillar recovery are 10 tonnes (12 tons/manshift) for cut-and-fill and 47 tonnes (52 tons/manshift) for VBM (Stewart, 1988). The impact of the difference in productivities on a time-based cash flow analysis would be significant.

The major disadvantage of VBM is its inability to follow irregular ore geometries, and the requirement for some overhead pillar protection to the drilling horizon. Ore recovery has been reduced by as much as one third in some stopes. Cut-and-fill mining will normally provide greater ore recovery due to its inherent flexibility.

A typical breakdown of direct mining costs for VBM is as follows (Ludwig, 1985):

Development:	22.1%

Labour:	20.7%
Backfill:	15.2%
Drill and Blast:	14.0%
Equipment repairs and maintenance:	11.3%
Sample drilling:	6.8%
Explosives:	5.4%
Miscellaneous:	4.5%

In terms of manhours, cut-and-fill will require about 60% more manhours than VBM to mine a given tonnage. The distribution of manhours by activity is as follows:

Activity	Cut-and-Fill	VBM (ramp access)
Production	37.1%	41.5%
Ground Support	19.6%	9.6%
Miscellaneous	19.2%	14.1%
Maintenance	11.2%	14.9%
Development	8.5%	13%
Fill Preparation	4.4%	6.9%

It should be noted that although a percentage given for Cut and Fill may be lower than the corresponding number under VBM (such as for fill preparation), the actual time required for the task may be higher because VBM requires less time overall to mine a given tonnage.

A critical component of the economics of VBM stoping is dilution. Potential sources are wall rock failure and sand from a neighbouring block. Experience at Thompson has shown that hangingwall failure is the predominant source of dilution. Statistics from ground control studies of VBM blocks is summarized as follows:

Hangingwall failures	31%
Back failures	11%
Footwall Failures	7%
No failures	51%

Total database = 126 blocks, mined 1981-1987

Hangingwall slough is normally approximated through observation of the stope walls when mining has been completed, and infrequently with downhole extensometers or cables. Estimations typically range up to 3.6 m (12 ft.) with two extreme values of 6.1 m (20 ft.) and 7. 6 m (25 ft.) The actual amount of rock missing off the walls is a judgement call which is difficult to make, so these figures should be taken as approximations only.

Hangingwall failure has been correlated with a number of parameters in efforts to determine its contributing factors, including powder factor, rock type, block dimensions and rock quality. Undoubtedly all of these will have some effect on dilution. The strongest correlation appears to be with rock type, particularly with the presence of a biotite schist in the hangingwall. This schist ranges in thickness from 1.5 - 7.6 m (5 to 25 ft.). In a total of 94 blocks with a schist hangingwall, failure occurred 34% of the time at an average slough of 2.1 m (7 ft.). Other rock types occur much less frequently and result in less hangingwall slough when they do occur.

When backfill from adjacent stopes is a source of dilution, the problem is easy to see but almost impossible to quantify since the volume of sand cannot be measured. No data is currently available on dilution by backfill.

Dilution is a serious problem in VBM stoping because VBM workplaces don't have a built-in ability to sort and store dilution as it is recognized. Some workplaces have limited storage for waste rock, but normally everything in the drawpoint is mucked as ore.

The economic consequences of dilution can be illustrated with the following example. Using the direct mining costs (including development) costs quoted by Ludwig (1985), and assuming an ore grade of 2.7% Ni, the following two cases represent identical mining costs per pound of nickel:

Cut and Fill: 10% dilution (assumed)

VBM: 36% dilution (calculated)

Hence, the choice between VBM and Cut-and-Fill must also consider realistic estimates of the expected dilution. The economic advantages of VBM can be quickly lost if dilution is not controlled.

4.0 VCR STOPING AT HOMESTAKE MINE

4.1 Introduction

Mining at the Homestake gold mine in South Dakota, USA, has provided an opportunity to evaluate VCR performance in both wide, large tonnage stopes and narrow small tonnage stopes. The large variation in orebody thickness and dip create difficulties quite similar to those encountered at the Thompson Mine, thereby permitting comparison of the two operations.

VCR mining was first introduced to the Homestake Mine in 1977, and by 1983 had become a fully integrated mining method. Previously, blasthole and cut-and-fill had been the dominant methods. However, cut-and-fill mining was plagued by steadily increasing costs and declining productivities due to its labor intensive nature. Conversion to blasthole stoping had been successful in the upper mine levels due to a competent hangingwall, but ground conditions deteriorated as mining went deeper. At deeper levels the Homestake formation, a sideroplesite-quartz schist which hosts all mineralization, is highly fractured with an average unconfined compressive strength of 19,000 psi (Orr, 1983). Excessive caving occurred at hangingwalls exposed by blasthole stoping. VCR mining has proven successful in this environment due to the wall support which it provides during production, its low cost and its high productivities.

The Homestake orebody is comprised of Precambrian metasediments (quartzites, phyllites and schist) which have been complexly folded into a series of synclines and anticlines. Ore zones are replacement pipes oriented parallel to fold plunges and range in shape from tabular blocks to small clusters or pods. They have the following characteristics (Orr, 1983, Haptonstall, 1984):

Ore width:	3.1 - 31 m (10 - 100 feet)
Ore dip:	50 - 80 degrees
Strike length:	9.2 - 152 m (30 - 500 feet)
Average grade:	0.16 oz/ton gold
Deepest mining depth:	2257 m (7400 feet)
Average mining depth:	1479 m (4850 feet)

Total mine production has progressively increased from 1.45 million tonnes (1.6 million tons) in 1978 to 2.2 million tonnes (2.4 million tons) in 1988 (Morgan, 1989).

4.2 Development

VCR stoping at Homestake requires an orebody dip greater than 60^{o}. Orebodies with flatter dips have not been mined successfully at Homestake because ore will not flow naturally along the footwall.

Overcut drill drifts are driven in ore parallel to strike. Minimum drift size is 3.6 m (12 feet) wide by 11 feet high to accommodate ITH drilling. If ground conditions permit, wider overcuts are driven to accommodate parallel drilling (Figure 6-12). 1.8 m (Six feet) of extra drift width is required in the footwall to drill parallel holes along the footwall. These "cleaner holes" provide for a smoother footwall and more efficient extraction (Andrews, 1984). Normally ground conditions do not permit full width overcuts, and fan drilling is required in these situations.

In very wide orebodies, transverse drill drifts are developed from a 3.1 m x 3.1 m (10 ft. x 10 ft.) access drift driven in waste along the hangingwall contact (Figure 6-13). Primary stope lengths of 9.2 m (30 feet) are mined out between pillars with strike length of 6.1 m (20 feet) minimum. Pillars are recovered following backfilling of primary stopes (Orr, 1983).

Undercut drifts are driven in ore such that the vertical distance between the overcut and undercut does not exceed 46 m (150 feet). The undercut is driven with a back height of 3.6 m (12 ft.) for a single cut, 7.3 m

(24 ft.) for a double cut. The double cut is driven to reduce the length of ITH holes required and to reduce damage to drawpoint brows from initial VCR blasting.

The haulageway is driven in the footwall parallel to orebody strike with dimensions of 3.1 m x 3.1 m (10 ft. x 10 ft.) Drawpoints are driven off the haulageway at a 60^{o} angle, and are spaced 11 - 12 m) 35 - 40 feet apart. These drawpoints are driven to intersect the undercut, and also have dimensions of 3.1 m x 3.1 m (10 ft. x 10 ft.) Length of drawpoint scrams vary from 12 m (40 feet) for LHD's to 7.6 m (25 feet) for the Atlas Copco Cavo 320 (Muir, 1988).

Normal stope dimensions are 46 m (150 feet) high, with widths varying from 4.6 - 30 m (15 - 100 feet) and strike length varying with orebody geometry.

4.3 Production Drilling

Drilling at the Homestake Mine utilizes ITH drill rigs and 165 mm (6.5 inch) diameter holes. Mine air pressure is boosted to 250 psi for the ITH hammer. Drill patterns vary depending on rock density and integrity. In the upper portions of the mine where rock structure is competent, hole spacing is 2.4 m x 2.4 m (8 ft. x 8 ft.) In deeper areas where rock is both weaker and more heavily fractured, a hole spacing of 3.1 m x 3.1 m (10 ft. x 10 ft.) has proven effective. Wherever possible, holes are drilled parallel to the hangingwall and footwall contacts.

At least 3 breakout holes are required in the undercut from each ring to produce effective fragmentation. In narrow stopes, this will often require a reduction to 1.8 m (6 ft.) spacing x 2.4 m (8 ft.) burden.

Hole deviation is caused by improper alignment, excessive feed pressure and geologic structure. All breakthrough holes are surveyed to check for deviation. In angled holes, excessive deviation usually results at depths greater than 46 m (150 feet). Measurements have indicated average deviation of 3% and maximum deviation of 8% (Mitchell, 1980).

4.4 Loading and Blasting

All holes are measured with a marked cable prior to loading, as well as the void above the muckpile. A minimum of 2.1 m (7 ft.) average void below each hole is required for proper expansion of the blasted ore.

Holes are blocked at 1.5 m (5 ft.) above back height with a wedge shaped block, along with drill cuttings and/or crushed gravel (figure 6-14). Each hole is loaded with a 27 kg (60 lb.) charge at a burial depth of 2.0 m (6.75 feet) above back height. Explosives used at Homestake include TNT-based water gels, emulsions and nitro carbonitrate water gels. (Mitchell, 1980). Detonation is via a 0.45 kg (1 lb.) cast primer with ms delay cap insert. The hole is then stemmed with 1.5 m (5 feet) of fine-grained sand and filled with water. This procedure provides excellent confinement and prevents hole rifling during detonation. Holes are tied in with E-Cord and initiated electrically.

Holes are delayed to fire a set of "break raise" charges on the first delay (Figure 6-15). Other holes follow in concentric rings around the initial holes, causing them to either crater downward or to slash into the void. For large blasts, a second blast is shot which consists of slashing a V-pattern into the void created by the

first blast, as shown in the figure. The maximum charge weight per delay is set at 225 kg (500 lbs.) to keep peak particle velocity less than 13 mm/s (0.5 inches/second).

VCR mining yields advance rates of 2.4 - 3.1 m (8 - 10 ft.) per blast at Homestake. Normal blasting is continued until 7.6 - 9.2 m (25 - 30 feet) of crown pillar remains in the stope. Any successive lifts would risk failure of the crown, so the remaining holes are loaded with multiple decks and shot in one blast. Standard 27 kg (60 lb.) charges are loaded in each deck with 1.5 m (5 ft.) of fine sand stemming between charges. Since both the stope void and overcut provide expansion room for the crown pillar blast, various blasting patterns have proven successful. Firing times between charge decks are normally delayed from 25-50 milliseconds (Andrews, 1984).

4.5 Mucking

Mucking of ore at stope drawpoints is carried out with electric and diesel powered 2.7 m^3 (3.5 cu.yd.), 1.5 m^3 (2 cu.yd.) and 0.765 m^3 (1 cu.yd.) LHD's, and with Atlas Copco Cavo 320's. The use of larger LHD's is hindered by the 2.4 m x 2.4 m (8 ft. x 8 ft.) drift size on most main levels. Diesel LHD's are most efficient in large stopes and areas where drifts connect multiple stopes. Cavo 320's work well in isolated small block stopes, where they consistently muck an average of 2990 tonnes (3,300 tons) per month (Muir, 1988). Muck is generally trammed to 2.1 m (7 ft.) diameter ore passes located along the main haulageway. Oversize at the grizzlies is broken with hydraulic hammers.

Once crown pillar blasting is complete, stopes are mucked empty. A remote controlled LHD is used for the final cleanup of ore left in the stope after conventional mucking. Empty stopes are backfilled as soon as possible to re-establish wall support. Bulk backfill consists of 15:1 sand:cement ratio.

4.6 Productivity and Performance Comparisons

A breakdown of stoping costs for VCR mining at Homestake is shown in Figure 6-16. Direct mining costs, including drilling and blasting, account for 30% of the total while diamond drilling and development account for 21.1%.

A comparison of direct mining costs and productivities for all mining methods at Homestake has been carried out for the years 1978-1983 (Orr, 1983). Mining costs have been normalized by year to the cost of blasthole stoping, then averaged for this 6 year period to remove the effects of economic fluctuations. Productivities have simply been averaged over the 6 year period.

Mining Method	Productivity tonnes/manshift (tons/manshift)	Normalized Direct Mining Cost per ton
Vertical crater retreat	27 (30.0)	1.74
Blasthole	30 (33.0)	1.00
Shrinkage	16 (18.0)	2.23
Overhand cut-and-fill	13 (14.8)	2.53
Mechanized cut-and-fill	18 (20.1) (82, 83 only)	2.36 (1983 only)

The productivity for VCR stoping is competitive with blasthole stoping and much superior to the other more manually intensive methods. The direct mining cost for VCR stoping is greater than that for blasthole, but less than the other methods. Development costs have not been considered here, but will be less for VCR.

VCR stoping is normally only attempted in large ore blocks. Because of its many advantages over cut-and-fill mining, 5 trial small block VCR stopes of between 13,600 (15,000) and 59,000 (65,000 tons) were mined out on a test basis (Muir, 1988). It was found that both productivity and cost/ton were comparable to those for mining of large blocks with VCR. However, dilution was much higher in small block VCR, as can be seen in the following comparison:

Dilution	Small block VCR:	44%
	Large block VCR:	25%
	Mechanized cut-and-fill:	15%
	Overhand out-and-fill:	10%

The total mining cost per ton and productivity for small block VCR was also compared with other mining methods, as shown below (Muir, 1988):

Mining Method	Productivity (tons/manshift)	Normalized Total Mining Cost Per Ton
Small block VCR	30 (33.5)	1.00
Large block VCR	32 (35.0)	1.05
Mechanized cut-and-fill	27 (29.3)	1.37
Overhand cut-and-fill	17 (18.4)	1.53

The comparisons show that VCR is a viable method in small ore blocks if extra dilution can be tolerated. Control of drillhole deviation is therefore critical to its successful application. Ground support requirements are much less than for either form of cut-and-fill stoping.

Studies at Homestake have compared the variation in total mining cost per ton with stope tonnage. Since total development costs for a large block or a small block VCR stope are quite similar, large blocks are more economically viable. Cost comparisons showed a 9 per cent increase in total mining cost per ton as stope tonnage dropped from 118,000 (130,000) to 68,000 (75,000 tons), and a 19 per cent increase from 118,000 (130,000) to 18,000 (20,000 tons).

5.0 VCR STOPING AT CAROLUSBERG MINE

5.1 Introduction

Mining at the Carolusberg Mine of the O'Okiep Copper Company in South Africa provides a good example of VCR stoping in a heavily-jointed orebody subjected to very high horizontal stresses.

Production from the Deep Ore Zone at Carolusberg commenced in 1983. The initial mining method used was sublevel open stoping, with mass-blasting of rib pillars into adjacent open stopes. This method was unsuccessful due to excessive caving of the open stopes, including failures of stope backs, rib pillars, crown pillars and drawpoints. The main causes were high horizontal stresses and intense jointing in the orebody (Bryant, 1986). VCR stoping was tested, and found to provide sufficient wall support to control ground failures. Other advantages of VCR included less sublevel development with the large diameter holes, less redrilling since these holes weren't as prone to damage, and safer working conditions for mine personnel. Mine production is currently at a rate of about 4,000 tonnes/day.

The deep ore zone can best be described as a massive pod-like orebody with lenticular wings (Figure 6-17). Wall rocks are hard competent gneiss and granite, with some quartzite and schist. Dimensions of the main ore pods are as follows:

Ore Zones	Height	Dimensions in Plan
South pod	650 m (2100 ft.)	120 x 150 m (393 x 492 ft.)
North lens	440 m (1400 ft.)	30 x 150 m (98 x 492 ft.)
West lens	235 m (770 ft.)	30 x 100 m (98 x 328 ft.)

Rock mechanics data for the Carolusberg Deep Ore Zone are summarized as follows (Langley-Cook, 1985):

Rock Unit	UCS (MPa)	Description
Ore (diorite, hypersthenite)	158-170	heavily jointed
Wall rocks (granite, gneiss)	190-220	hard, competent
Depth (m)	**Max. Horizontal Stress (MPa)**	**Vertical Stress (MPa)**
850 Level (top of Deep Ore Zone)	50	23
1550 Level (bottom of Deep Ore Zone)	92	43

5.2 Development

Stope development at Carolusberg involves drill drifts spaced at 50 m vertical intervals. Initially, double drill drives were developed along both stope boundaries to enable drilling down the planned walls. However, this weakened the stope corners and induced overbreak where high stress concentrations were present. At present, single drill drifts are driven along the stope centre line. Drift dimensions are 4.2 x 3.4 m (Marvin, 1988).

A single drawpoint is normally developed for each VCR stope. The ore is undercut in a T-shape as shown in Figure 6-18, thus providing an initial breaking face for crater blasting. A small raise is also developed to establish a drawpoint brow and protect the drawpoint from the initial blasting shock. Typical haulage drift dimensions are 4.2 x 3.4 m.

All access tunnels, drawpoint haulages and drill drifts are developed parallel to the direction of maximum principal stress wherever possible. If this is not possible, extra support is installed. Normal ground support involves 1.8 and/or 3 m long swellex bolts and wire mesh on walls and backs. Tunnels are driven with a very slightly arched crown to reduce stress concentrations around openings.

Drawpoints are not developed until the stope is ready for production. During mining, drawpoints are subjected to very high stress levels, abrasion from the flow of broken ore and concussions from secondary blasting. Extra ground support is therefore provided, including a heavy pattern of 3 m swellex bolts with plate washers, and a concrete bullnose at the drawpoint brow.

A stoping plan is shown in Figure 6-19. Stopes are vertical, with walls generally consisting of ore and/or cemented backfill. Stopes are laid out in a 20 m x 20 m grid pattern, or with modified dimensions to accommodate ore geometry at waste contacts. The maximum stope height is 50 m. Access ramps between levels are developed in waste rock, with a maximum gradient of 20 per cent.

5.3 Production Drilling

Drilling at Carolusberg utilizes ITH drill rigs and 165 mm holes. Mine compressed air is boosted to 1.7 MPa before being fed to the ITH hammer. Water is injected into the air stream to suppress dust.

Down holes are collared to 1.5 m depth using a 177 mm bit. A plastic standpipe is inserted to prevent drill cuttings from re-entering the hole. The hole is then completed with a 165 mm bit, and sealed with a conical wooden plug.

Production holes are ring drilled from the drill drifts, as shown in Figure 6-18. The planned maximum hole spacing is 3.3 x 3.3 m.

All production holes are surveyed with a down-hole instrument. Hole locations are plotted on plan at 2.5 m vertical intervals, and extra drilling is done if hole deviation has resulted in excessive spacing. Maximum lengths for production holes have been about 70 m.

5.4 Loading and Blasting

Prior to loading, the void and hole lengths are measured with a weighted marked rope. Holes are plugged with a conical wooden plug, aggregate and sand. Slurry explosive is added to fill 0.8 m of the hole, yielding a 24 kg charge. The explosive currently being used is Energex 455, which has the following properties:

Velocity of detonation:	4500 m/sec.
Density:	1.4 gm/cc
Relative weight strength:	117
Relative bulk strength:	204

Initiation is via a 400 gm pentolite booster and millisecond electric detonator. Stemming consists of 2 m of sand.

The charge burial depth of 2.0-2.5 m has proven to provide optimal blast performance. Advances of up to 4 m per blast are normal, with good fragmentation. The high advance rate per blast is due to the caving out of a highly stressed zone above the crater blast. Powder factors average 0.34 kg/tonne for primary blasting.

Horizontal stresses tend to form stope backs into a gentle arch. Therefore, each blast is delayed so that holes at the highest point of the arch fire first, followed by the others in concentric rings around the first holes. When mining commences in a given stope, the drawpoint is mucked empty after each blast to enable visual inspection of the stope back. The back is monitored in this way until breakage is satisfactory.

During stope production, the level of broken ore is not allowed to drop more than 4 m below the stope back. This allows sufficient void space for each VCR blast and maintains wall stability. About 40% of the broken ore is mucked during blasting operations.

Normal VCR blasting is stopped when the crown pillar thickness has reached about 12 m. A 4 m wide slot is then developed at one end of the stope, and the crown is slashed into this slot.

Once blasting has been completed, the remaining 60% of broken ore is mucked as rapidly as possible. Muck is carried out with 4 cu.m. LHD's, which deliver ore to one of several internal ore pass systems (Bryant, 1986).

The mine has encountered considerable problems with overbreak of stope walls and backs. This material is generally mucked for ore, but considerable effort is expended to blast oversize (30-40 percent of total powder consumption). Cable bolting of stope walls and backs had not been done as of 1987.

5.5 Backfilling

All stopes are backfilled when mucking has been completed. Backfill is designed to stand safely with exposures up to 40 m in width and 50 m in height. It consists of the following (Langley-Cook, 1985):

70% unclassified mill tailings (60% minus 74 microns);

24% dry tailings from the slimes dam;

6% reverberatory furnace slag.

This fill is sent underground as a 75 per cent solids mix so that minimal drainage is required in the stope. Backfill strength exceeds 800 kPa after 28 days.

Backfill barricades in the drawpoints consist of mass concrete walls. Other accesses are barricaded with triple brick walls. During backfilling operations, samples are taken on surface and underground for strength testing after selected curing times.

6.0 MODIFIED VCR/LONGHOLE STOPING AT CSA MINE

6.1 Introduction

Mining at the CSA Mine in New South Wales, Australia, uses a system which utilizes the advantages of both VCR and longhole blasting to optimize production. Since the primary disadvantage of VCR blasting is damage to stope walls and hence increased dilution, the CSA Mine slashes perimeter holes into the stope void using longhole techniques. This void is initially created by VCR blasting a slot raise up the middle of the stope while progressively slashing into this slot as it advances upward. The advantages of this stoping technique include better control of fragmentation, increased productivity and major reductions in secondary blasting costs (Munro, 1985).

Production at the CSA Mine commenced in 1964. It produces copper, lead, zinc and silver at a rate of 850,000 tonnes per year. In 1985, the historical head grade of 1.65% Cu, 0.7% Pb and 2.23% Zn was not economically viable, so efforts were made to increase the copper grade to at least 2%.

The CSA orebodies are lenticular in shape with variable thicknesses and dips between 75 and 80 degrees. The host rock is a steeply dipping siltstone. Dilution normally originates from a 4 to 5 m thick cleavage zone of black chlorite shears at the hangingwall and footwall contacts. Any overblasting causes significant sloughage of the stope walls. The sharp ore/waste contact means that any wall failure has a significant impact on head grade. A typical stope geologic section is shown in Figure 6-20.

The original mining method at CSA was cut-and-fill. The increased cost of ground control as ground conditions deteriorated, combined with falling copper prices in the 1970's, forced the change to longhole open stoping in 1978. Further optimization work has produced the present mining method.

Most stopes are mined longitudinally. A few transverse stopes have been mined in the wider ore zones.

6.2 Development

The main access ramp is driven at 14% with level intervals at 90 m. Typical dimensions for open stopes are 35-50 m height, 40-80 m length and 5-40 m width. Drawpoints are driven 15 m long at 60 degrees to the extraction drift in a herringbone arrangement. Drawpoint dimensions are 5 m wide x 4 m high.

Slot raises are developed by the longhole raise method (Price, 1983) which gives rapid advances under all types of ground conditions. The method is described in Figure 6-21, and involves full face firing of up to 9 charged holes. Holes are drilled at 165 mm diameter using ITH drill rigs. Hole deviation has reached 4%, but is commonly less than 2%. The geometrical design of the 10-hole pattern allows for re-arrangement of the pattern at different horizons following distortions caused by hole deviations.

Slot raises are positioned in the centre of the stope to minimize blast damage to the wall caused by the high powder factors in spherical blasting. Other types of raises are also developed using this method, including vent raises and ore passes. Longhole raising experience at CSA Mine is summarized as follows (Price, 1983):

Raise length developed:	52 m maximum
Advance per firing:	0.9 - 6.2 m
Explosives used:	ANFO and Tovex
Powder factors:	3.0 - 11.3 kg/tonne

A typical stope layout is illustrated in Figure 6-22. Stopes are undercut with a 2.5 m backslash to protect the drawpoint brow from production blasting and to decrease the possibility of blasthole freezing with multi-row firings. This undercut is drilled with a development jumbo and is taken out before production drilling begins. The improvement in drawpoint stability since the introduction of undercutting has reduced the support requirements at drawpoints.

6.3 Production Drilling

All production drilling is carried out with ITH hammer drills on Atlas Copco ROC306 drill rigs. Longhole raises are drilled with 165 mm bits whereas stope blastholes are 140 mm diameter. For both types of drilling the drill pipe is 115 mm OD with no stabilizers or guide subs. Air pressure is boosted to 1,700 KPa with portable compressors.

Initial experience with smaller diameter holes showed excessive deviation over 40 m lengths. The 140 mm holes are drilled with the same drill pipe as the larger holes to obtain a "stiffer" rod string. This results in an increased drilling accuracy. Hole surveys have shown that deviation usually results from error in hole bearing and not dip.

The current drilling pattern is 3 m burden x 3.5 m toe spacing. Wall holes are drilled 1.0 m from the footwall and hangingwall contacts to help prevent overbreak of stope walls. All production holes are drilled down from the drill drifts.

6.4 Loading and Blasting

The high pyrite and pyrrhotite content of the orebody has caused dust explosions during blasting operations. To minimize this hazard, a restriction of 40 milliseconds is placed on the maximum time delay between detonators. This places a major restriction on stope blasting practice. Lime dust and water sprays are also used to suppress propagation of dust explosions.

The loading arrangement for blasting of longhole raises is shown in Figure 6-21. Holes are measured, then blocked off with a wooden block and rope. Lime is placed at the bottom of the hole, then 1 m of sand stemming to seal the hole. About 5 m (15 ft.) of explosive charge is placed, with detonator, then 3 m (10 ft.) of sand stemming. It is very important that each hole in a raise round be charged at the same height (Price, 1983).

As the raise nears the top, the length of charge fired at one time has to be reduced or the top of the raise will crater upward, making it very difficult to take the next round. Advance rates of 6 m are commonly taken until a 10 m crown remains.

The first three holes in the cut are charged with Tovex Extra and ANFO combined. This produces a greater velocity of detonation than the surrounding stripping holes which are charged with ANFO only. This ensures that the cut area is partially free of ground when the outside holes fire. Wet holes must be charged with a water-gel explosive only.

Charge initiation is done with a non-electric system consisting of slider primers, primadets and E-Cord. The delay pattern used is shown in Figure 6-21 .

Stope blasting utilizes column charging of 140 mm holes to slash into the void created by the slot raise. Normal stope blasting uses ANFO or 30% diluted ANFO in column charges or Tovex water-gel in wet holes. Wall holes are loaded with diluted ANFO (locally mixed, called ANFOPS-30), and this is quite effective at reducing blast damage to stope walls.

Column charge initiation utilizes Ensign-Bickford HD-Nonel detonators, slider primers and Dupont RX Primaline in the hole. E-cord is used for surface trunklines.

The standard idealized firing sequence for stope blasting is shown in Figure 6-22. It utilizes underhand bench blasting to slash into the stope void. Previously, the longhole raise was advanced well ahead of production blasting, but this resulted in excessive oversize due to failure of the brow at the raise bottom. To retain control over fragmentation, it was decided to utilize a system of multi-row underhand benching, where these brows are blasted before they break away on their own. It involves firing to a stable arch to maintain back stability.

The normal practice involves keeping the longhole raise 5 m (15 ft.) above the slash blasts. As the raise and slot progress upwards, the bottoms of the main rings are progressively brought in. The raise is advanced until a 5 m (15 ft.) crown remains, then the entire slot is mass fired to breakthrough (Munro, 1985). Subsequent blasts are slashed into the slot void as shown in Figure 6-22.

6.5 Overall Performance

The adoption of this stoping technique has provided the following advantages:

1) Significantly reduced wall dilution, providing a head grade of 2.4% Cu in 1985 compared to 1.65% Cu using longhole open stoping.

2) Control of fragmentation which has dramatically reduced secondary blasting costs. Permanent secondary blasters were eliminated from each shift.

3) Increased productivity. The stope is brought into production sooner due to more broken ore being generated as raise development progresses.

4) Dependence is taken off the initiation system since each row has its own free face. The effect of an out of sequence firing is minimized.

5) Charge weights are controlled by firing sections of holes in different firings without the need for deck charging.

6) Vibration levels are reduced by the low charge weight per delay.

7) The direction of firing is down and away from the stope edge, thereby reducing the amount of fly dirt and lessening the cleanup after each firing.

The major disadvantage of the method is the increased manpower required due to the greater number of firings involved. On average, there are three to four firings per week varying from 2,000 to 5,000 tonnes each.

The practice of using diluted ANFO in larger holes 140 mm (5.5 inch) for slashing has permitted a lower powder factor while still maintaining drilling accuracy with the stiffer drill string.

7.0 OVERALL PERFORMANCE COMPARISON

The VBM method has been successfully used at Thompson and at a number of other mines worldwide. A common thread between the mines is the use of VBM to convert from cut-and-fill methods to more productive bulk mining or, to convert areas experiencing significant ground control problems. Interestingly, though several mines were identified that were originally planned as VBM operations (e.g. Rubiales in Spain), few continued with the method beyond initial start-up.

Table 6-1 summarizes key statistics from the Thompson Mine and three other operations selected for comparison. Perhaps the most outstanding difference between Thompson and these other mines is average stope tonnage. Typically, Thompson will draw about 18,140 (20,000 tons) of ore from a stope compared, with the 90,700 (100,000 tons) drawn at the other mines. This emphasizes the application of VBM at Thompson to both wide and narrow orebodies, while at the three other mines, VBM tends to be used only in wider sections of their orebodies. Homestake did try mining narrow orebodies, but found that though productivities were comparable with wider orebodies, dilution (primarily a result of hangingwall blast damage) was very high. Thompson has, for the time being, accepted this trade-off between increased productivity and increased dilution associated with mining narrower orebodies.

Another area of difference is in the use of a shrink draw. The tendency at Thompson is to pull stopes empty after blasting each lift, whereas the other mines will operate a shrink draw to provide maximum hangingwall support during mucking. Though the benefit of a shrink draw is strongly endorsed at Thompson, the reality of operating a mature mine with many scattered working places, and poor access for trackless equipment (a function of the conversion) results in severe production demands from individual stopes.

At Thompson, the VBM method has been used in conjunction with crater blasting techniques. Ideally, this approach is suited for wide orebodies (greater than 25 ft.) with strong competent footwalls and hangingwalls. Thompson have been successful in applying the method in far from ideal circumstances, that is in narrow orebodies with weak hangingwalls, in order to achieve major productivity gains. The trade-off has, however, been increased dilution. Because of the grade of the Thompson orebody and historical nickel prices, this trade off has generally been acceptable. However, recent high nickel prices have substantially increased the cost of dilution, and the mine has embarked on a program to further investigate the causes and adjustments required to the VBM method in order to minimize the amount of dilution.

DAM/gc
2/852-1605

TABLE 6.1 AVERAGE STATISTICS

MINE	OREBODY CHARACTERISTICS					LAYOUT			PRODUCTION									BLEND DRAW	DILUTION		COMMENTS
	DIP (DEG)	WIDTH (FT)	R'WALL	ORE	STOPE	STOPE HEIGHT (FT)	STOPE LENGTH (FT)	DRAW	DRILL HOLE SIZE (INCH)	PATTERN	SPACING BURDEN (FT)	POWDER FACTOR	DEV (%)	STOPE TONNAGE (TONS)	VCR ADVANCE (FT)	BACKFILL			SMALL STOPE	LARGE STOPE	
THOMPSON CANADA	70	5-100	SCHIST, WEAK, FOLIATED	SULPHIDE COMPETENT	150	60	15 - 40 (LONG. RETREAT) UP TO 100 TRANVERSE	LONGITUD. F' WALL	6.5 6.5	FAN	6.5 BLASTHOLE 6 X 10 6.5 BLASTHOLE 5 X 7	1.05LB/TON	UP TO 5%	20,000	10	CEMENTED TAILINGS 4% CEMENT	NO		37.5*	25.5*	CONVERSION FROM CUT - AND - FILL MATURE MINE WITH NUMEROUS WORKPLACES. MAJORITY OF STOPING BY LONGITUDINAL RETREAT
HOMESTAKE (USA)	70	10-100	SCHIST FOLIATED	SULPHIDE COMPETENT	150	50	15-100	F' WALL	6.5	FAN	8 X 8 TO 10 X 10	1.2LB/TON COMPETENT ORE 0.8LB/TON IN FRACTURED ORE	3% UP TO 8%	100,000	8-10	CEMENTED TAILINGS 6% CEMENT	YES		44%	25%	CONVERSION FROM CUT - AND - FILL MATURE MINE WITH NUMEROUS WORKPLACES. DOUBLE PATTERN OF PRIMARY AND SECONDARY STOPES.
CAROLUSBERG SOUTH AFRICA	90 MASSIVE FOD	100 - 240	GNEISS COMPETENT	JOINTED COMPETENT	JOINTED DIORITE	165	65	WITHIN OREBODY	6.5	FAN	11 X 11	0.7LB/TON	N/A	70,000	15	CEMENTED TAILINGS 6% SLAG	YES		NONE	WALL AND BACK STAB. A MAJOR PROBLEM	CONVERSION FROM BLASTHOLE, EXCESSIVE SLOUGH USING BLASTHOLE OPEN STOPING. HIGH LATERAL INSITU STRESSES A PROBLEM.
CSA AUS.	75	15-130	CHLORITE SERICITE 15FT THICK	SULPHIDE COMPETENT	150	200	60	LONGITUD.	5.5 SLASHING 6.5 FOR SLOT RAISE	FAN	10 11.5	0.5LB/TON FOR SLASHING*	2% UP TO 4%	100,000	20 FT IN SLOT RAISE	CEMENTED TAILINGS 3-6% CEMENT	YES		N/A	10%	CONVERSION FROM CUT - AND - FILL MATURE MINE, SLOT RAISE VCR BLASTED, THEN LONGHOLES

* NOTES ESTIMATE FROM 1989 PRODUCTION FIGURES

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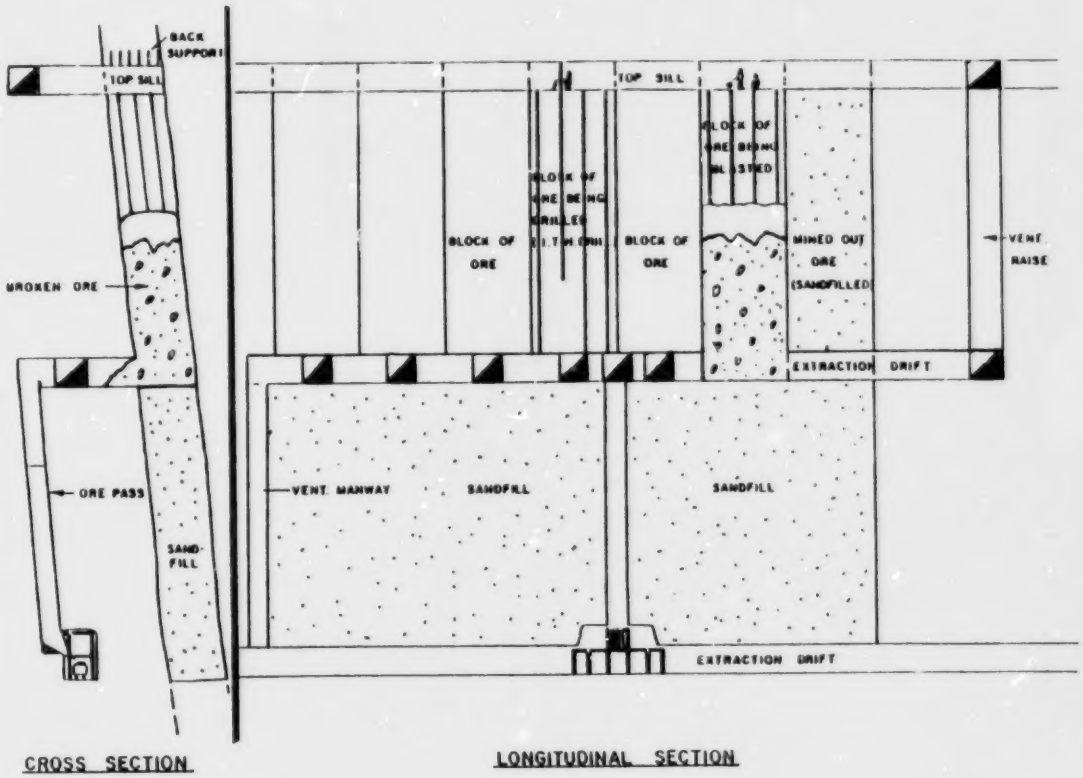
TYPICAL VBM STOPING ARRANGEMENT

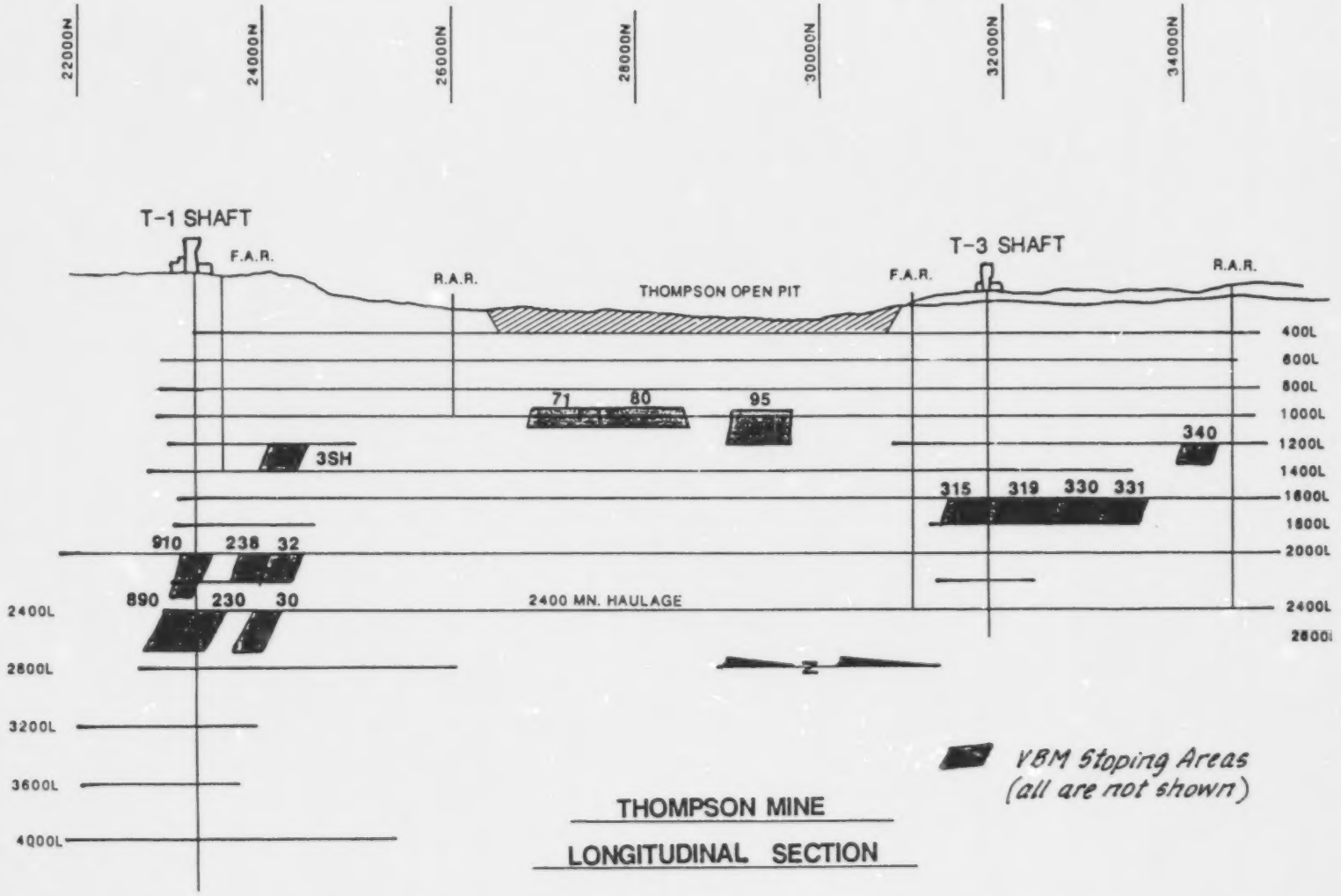
Figure **6-1**



PROJECT NO. 852-1605 DRAWN ..... REVIEWED ..... DATE Sept. '89

Reference: Doman, 1989

PROJECT NO. DRAWN REVIEWED DATE



THOMPSON MINE

LONGITUDINAL SECTION

Reference : Golder 1987

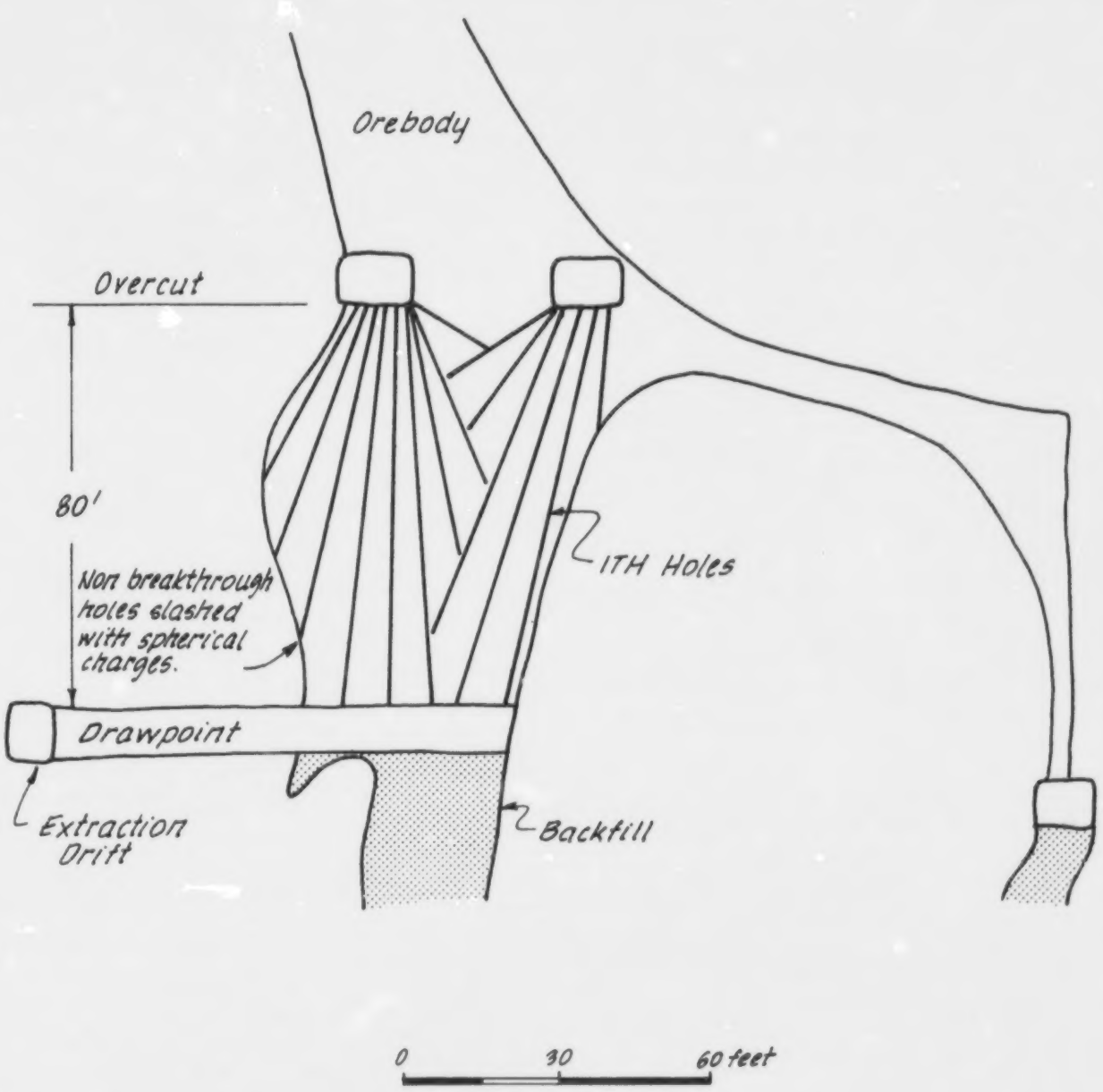
Golder Associates

THOMPSON MINE
LONGITUDINAL SECTION

Figure 6-2

EXAMPLE OF TWIN OVERCUT DRILL DRIFTS
THOMPSON MINE

Figure **6-3**

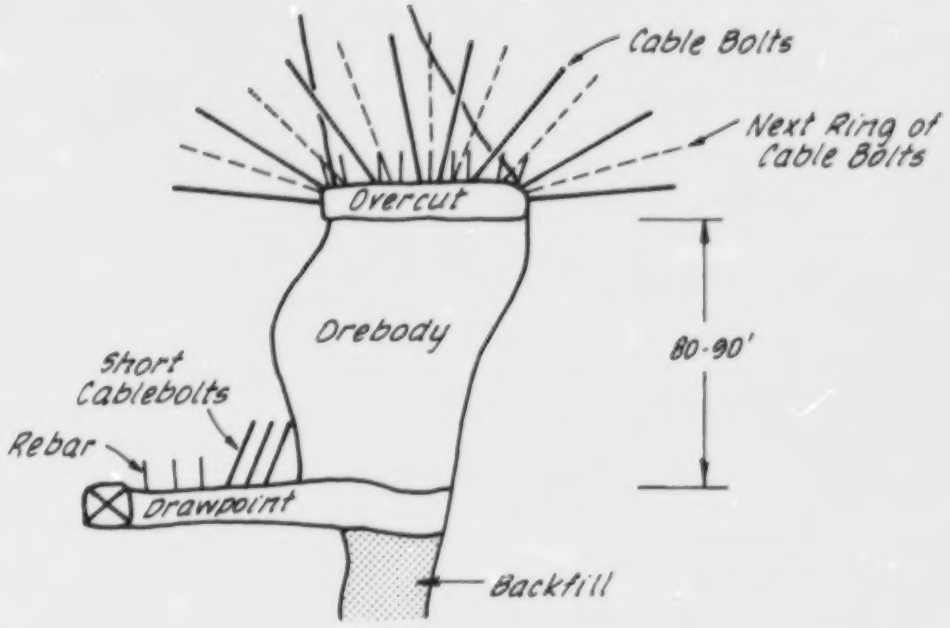


Reference: Inco Ltd.

PROJECT NO 852-1605 DRAWN - REVIEWED DATE Nov. '85

GROUND CONTROL IN WIDE OVERCUT DRILL DRIFT
THOMPSON MINE

Figure **6-4**



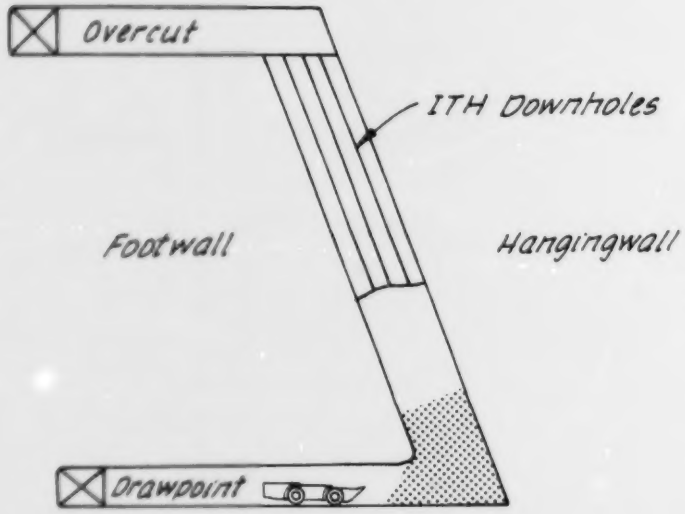
Not to Scale.

Reference: Inco Ltd.

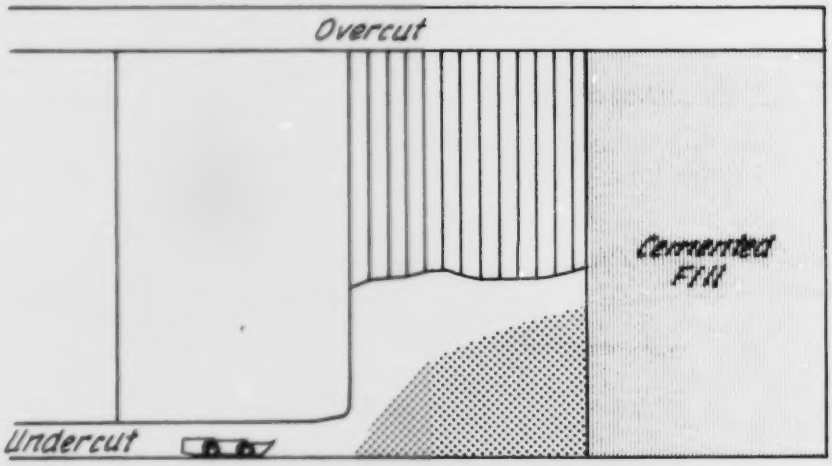
PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

VCR STOPING WITH PARALLEL HOLE DRILLING
THOMPSON MINE

Figure **6-5**



Cross-Section (Transverse Retreat)



Longitudinal Section (Longitudinal Retreat)

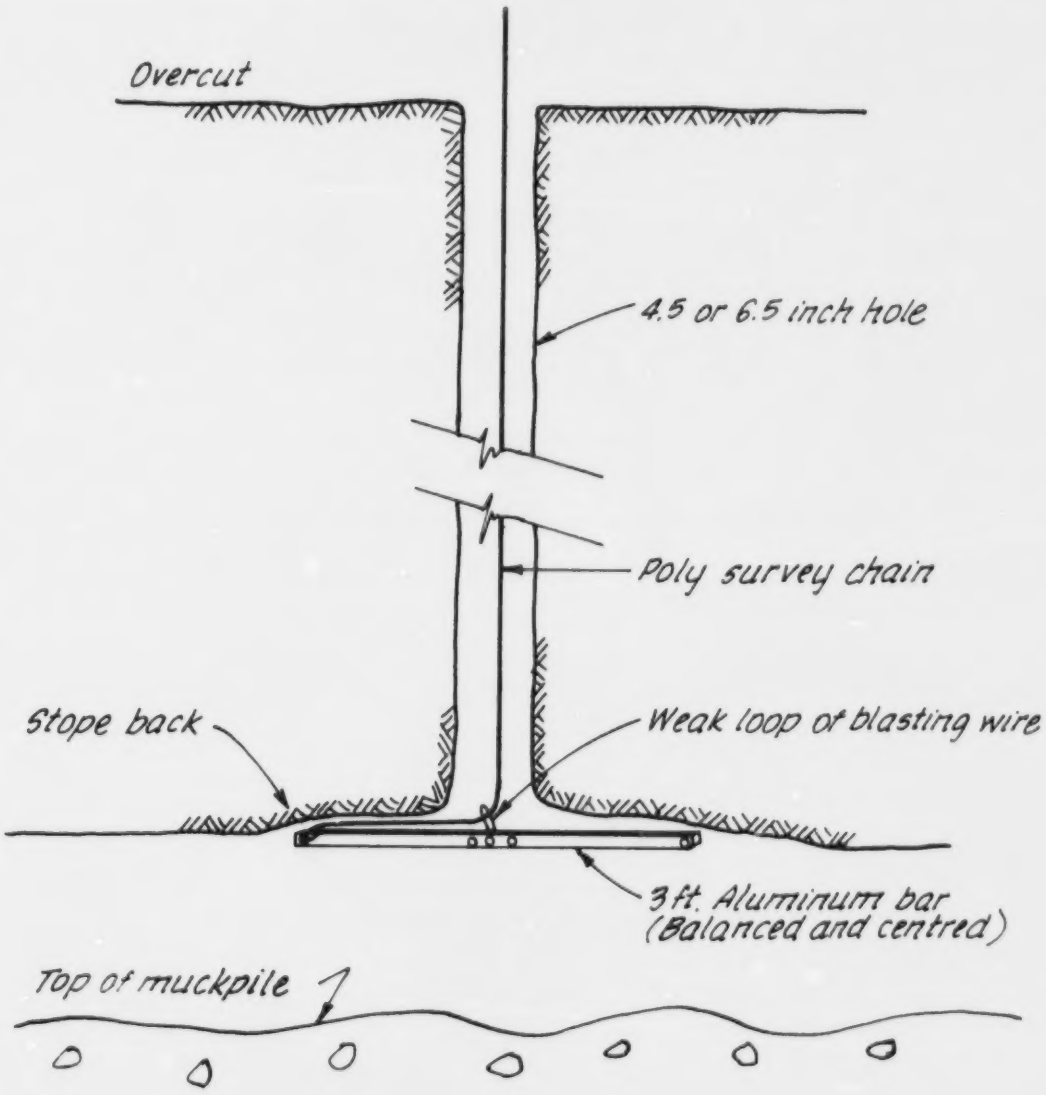
Not to Scale.

Reference: Inco Ltd.

PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

ITH HOLE MEASURING ARRANGEMENT

Figure **6-6**



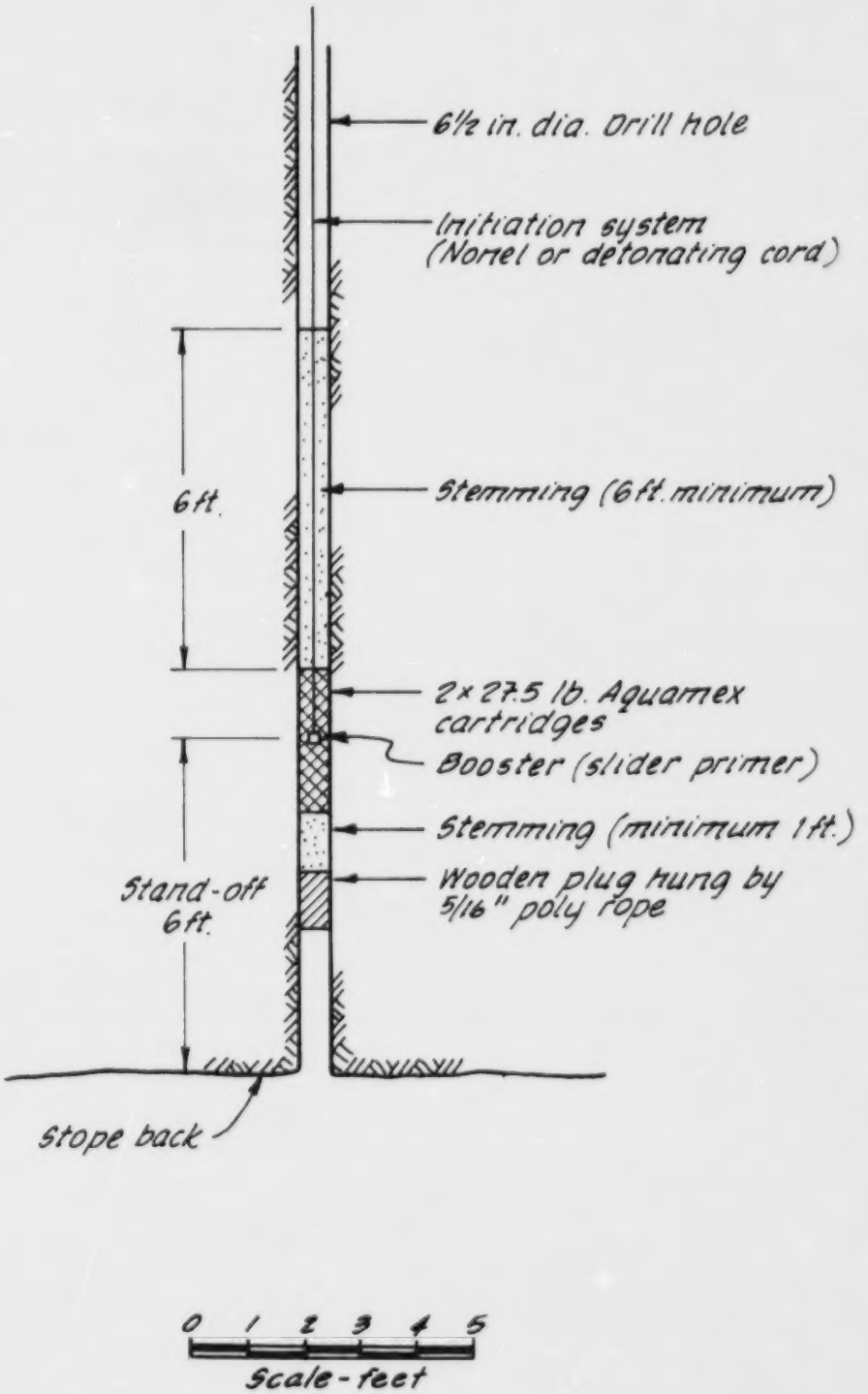
Not to Scale.

Reference : Doman, 1989

PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

CHARGE CONFIGURATION

Figure **6-7**

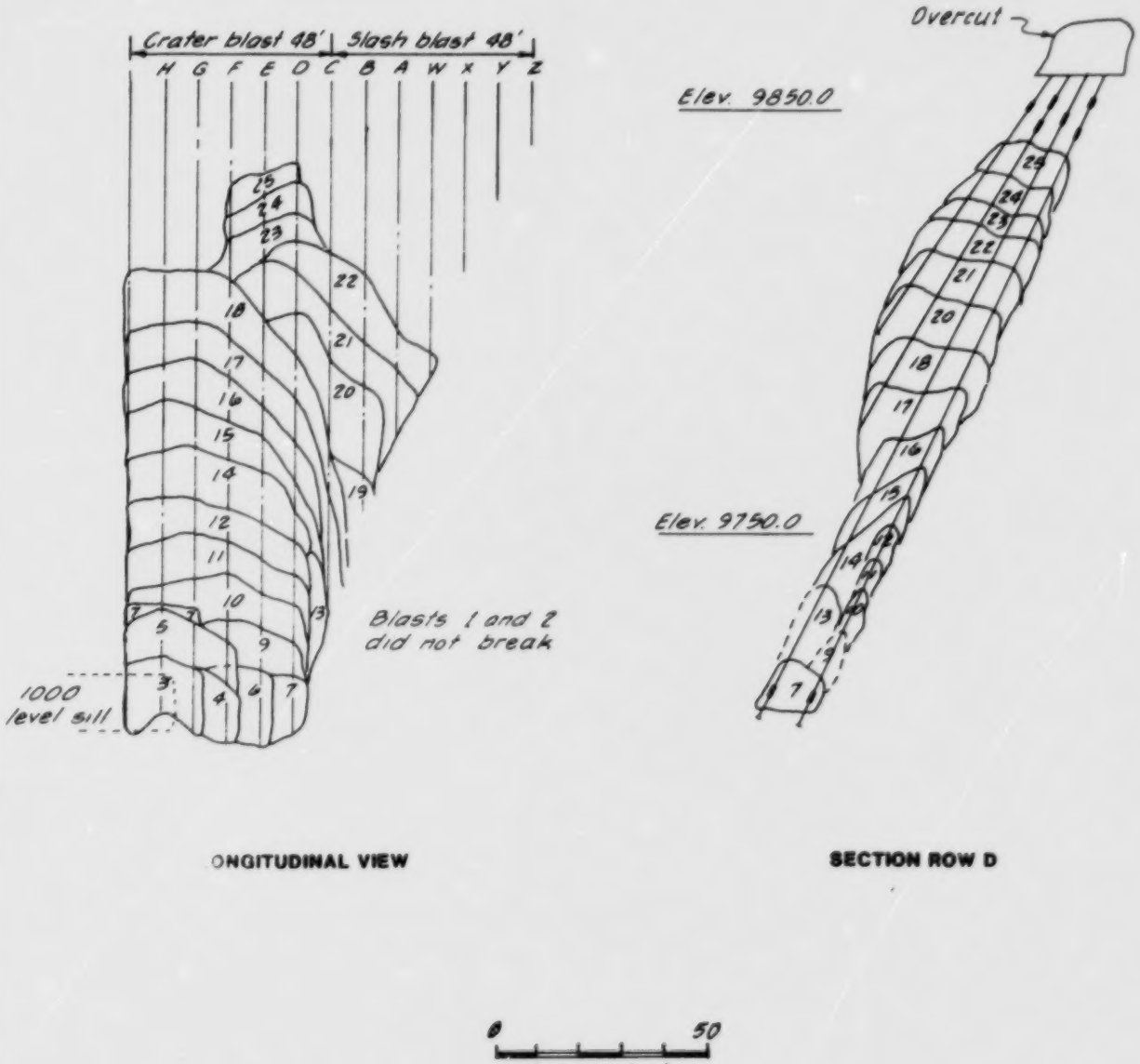

6½ in. dia. Drill hole
Initiation system (Nonel or detonating cord)
6 ft.
Stemming (6 ft. minimum)
2 x 27.5 lb. Aquamex cartridges
Booster (slider primer)
Stemming (minimum 1 ft.)
Stand-off 6 ft.
Wooden plug hung by 5/16" poly rope
Stope back
0 1 2 3 4 5
Scale - feet

Reference: Golder, 1987

PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

VCR AND SLASH BLASTING
THOMPSON MINE

Figure **6-8**



LONGITUDINAL VIEW

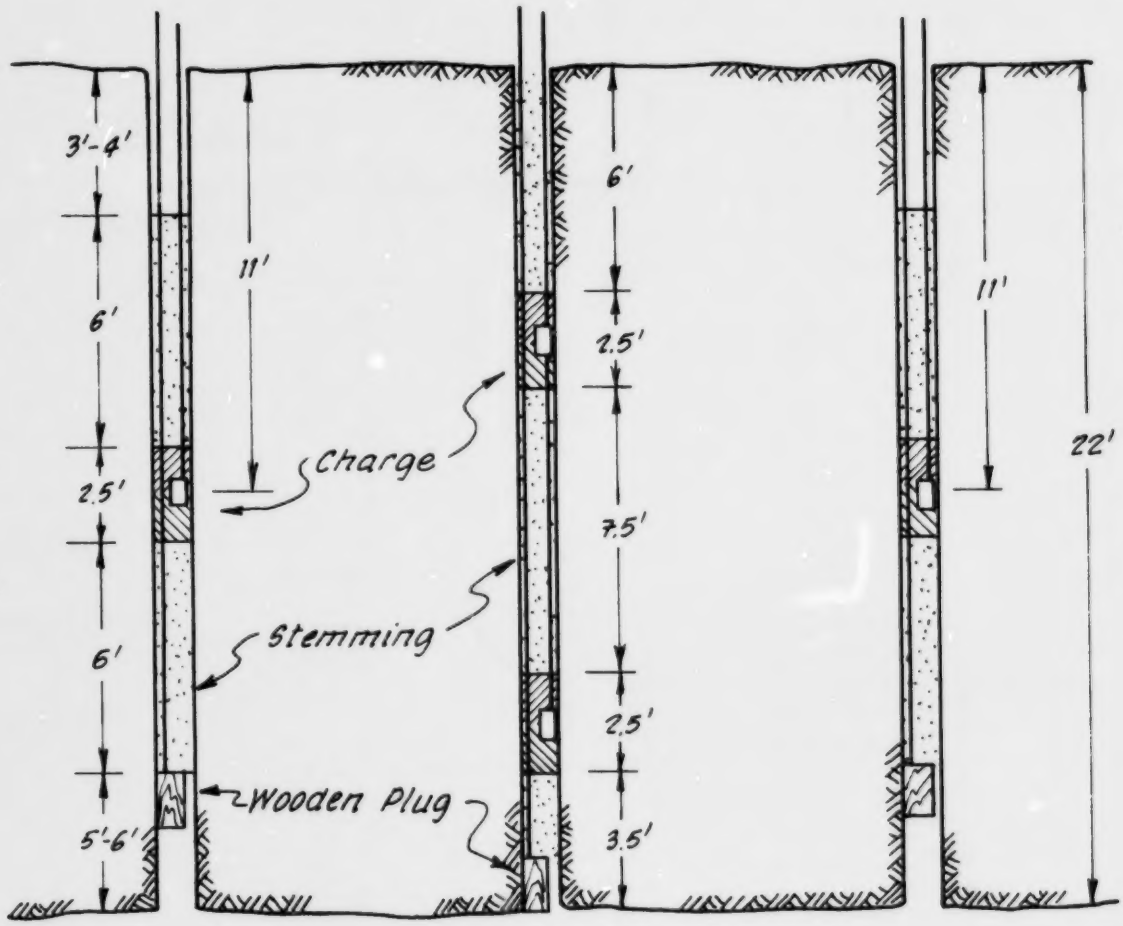
SECTION ROW D

Reference : Golder, 1987

PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

GENERAL ARRANGEMENT FOR FINAL BREAKTHROUGH BLAST

Figure **6-9**

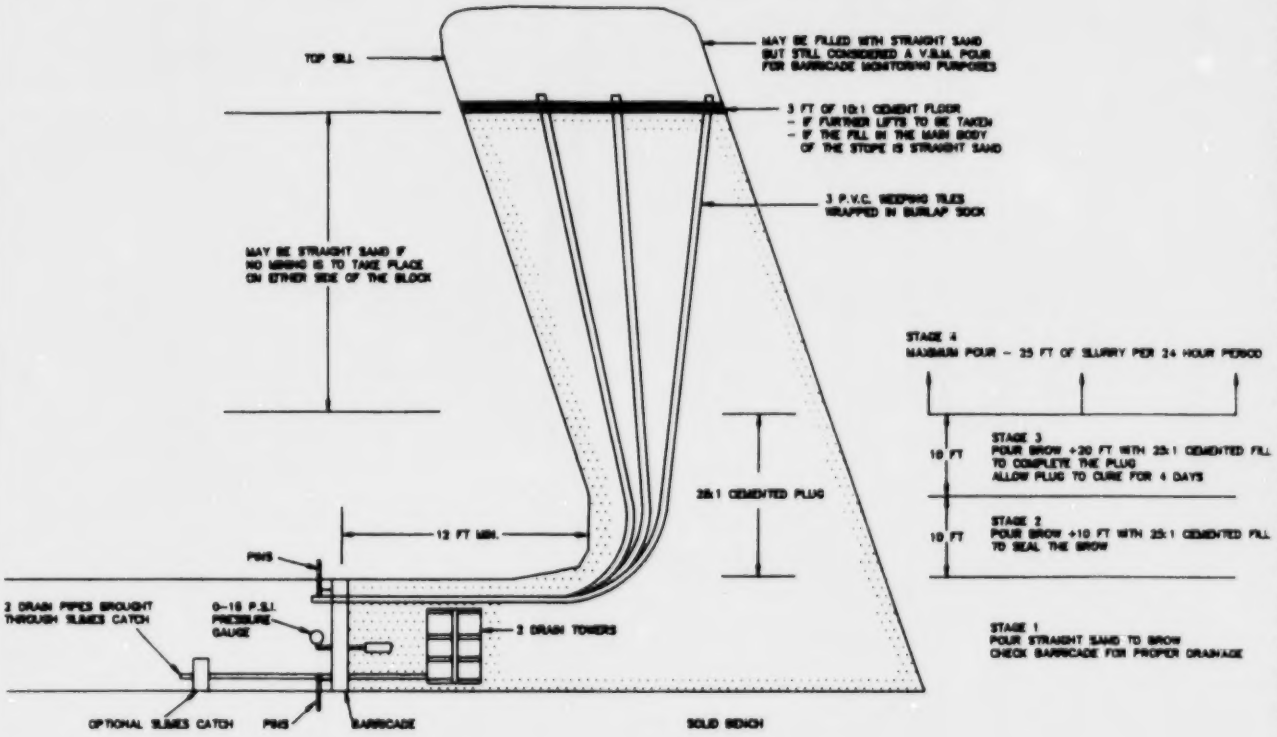


Not to Scale.

PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

GENERAL ARRANGEMENT FOR VBM BACKFILL POURS

Figure **6–10**


TOP SILL
MAY BE FILLED WITH STRAIGHT SAND BUT STILL CONSIDERED A V.B.M. POUR FOR BARRICADE MONITORING PURPOSES
3 FT OF 10:1 CEMENT FLOOR
– IF FURTHER LIFTS TO BE TAKEN – IF THE FILL IN THE MAIN BODY OF THE STOPE IS STRAIGHT SAND
3 P.V.C. WEEPING TILES WRAPPED IN BURLAP SOCK
MAY BE STRAIGHT SAND IF NO MINING IS TO TAKE PLACE ON EITHER SIDE OF THE BLOCK
STAGE 4
MAXIMUM POUR – 25 FT OF SLURRY PER 24 HOUR PERIOD
10 FT
STAGE 3
POUR BROW +20 FT WITH 25:1 CEMENTED FILL TO COMPLETE THE PLUG
ALLOW PLUG TO CURE FOR 4 DAYS
10 FT
STAGE 2
POUR BROW +10 FT WITH 25:1 CEMENTED FILL TO SEAL THE BROW
25:1 CEMENTED PLUG
12 FT MIN.
PINS
2 DRAIN PIPES BROUGHT THROUGH SLIMES CATCH
0–10 P.S.I. PRESSURE GAUGE
2 DRAIN TOWERS
STAGE 1
POUR STRAIGHT SAND TO BROW
CHECK BARRICADE FOR PROPER DRAINAGE
OPTIONAL SLIMES CATCH
PINS
BARRICADE
SOLID BENCH

Not to Scale.

NOTE:

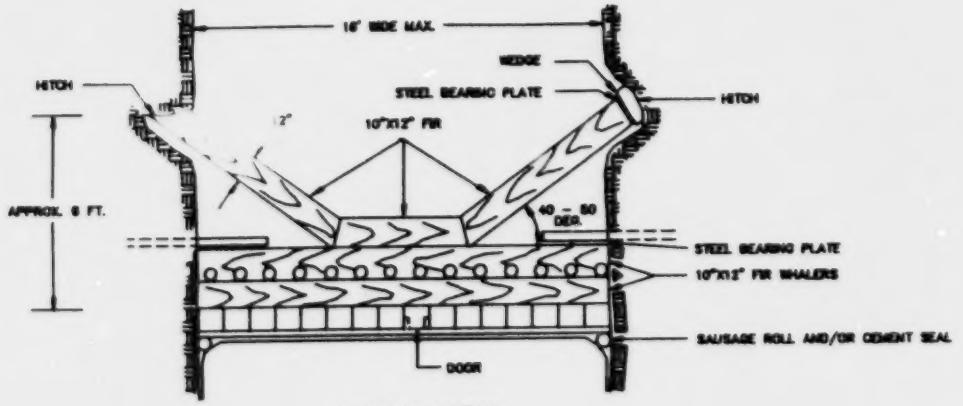
DRAIN PIPES ARE CURRENTLY NOT INSTALLED DUE TO PROBLEMS WITH SLOUGHING WALLS RIPPING OUT THE PIPES AND CREATING SAFETY HAZARDS TO PERSONNEL DURING INSTALLATIONS

PROJECT NO 852-1605 DRAWN — REVIEWED DATE Nov. '89

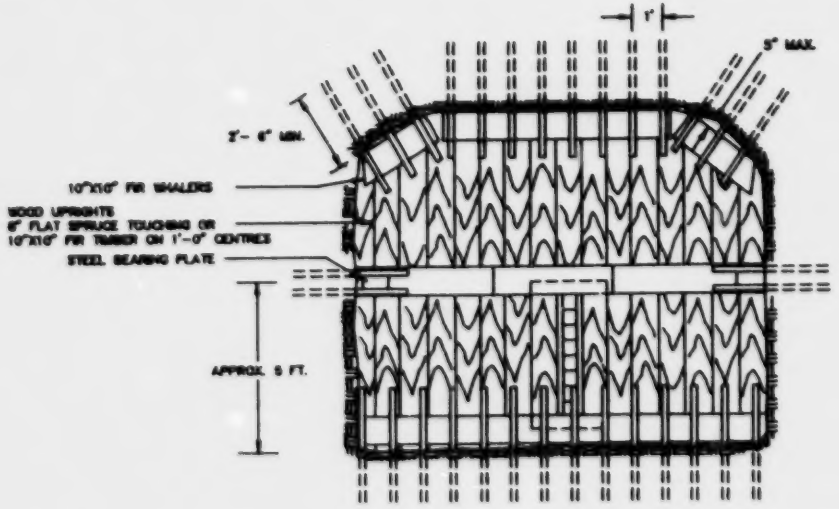
Reference: Inco, 1988.

PROJECT NO 852-1605 DRAWN — REVIEWED DATE Nov 1989

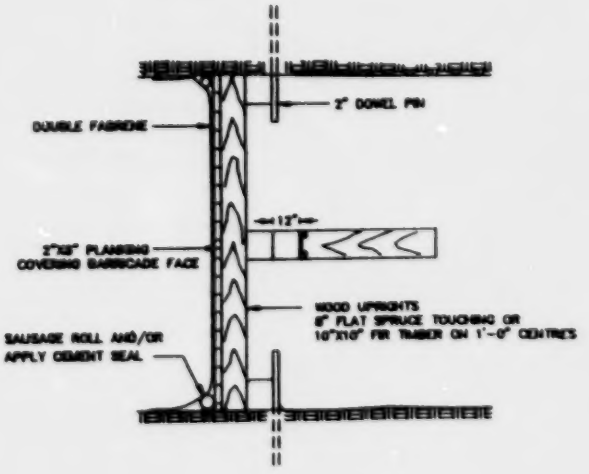
TYPICAL VBM BACKFILL BARRICADE



PLAN VIEW



FRONT VIEW



FRONT VIEW

Not to scale

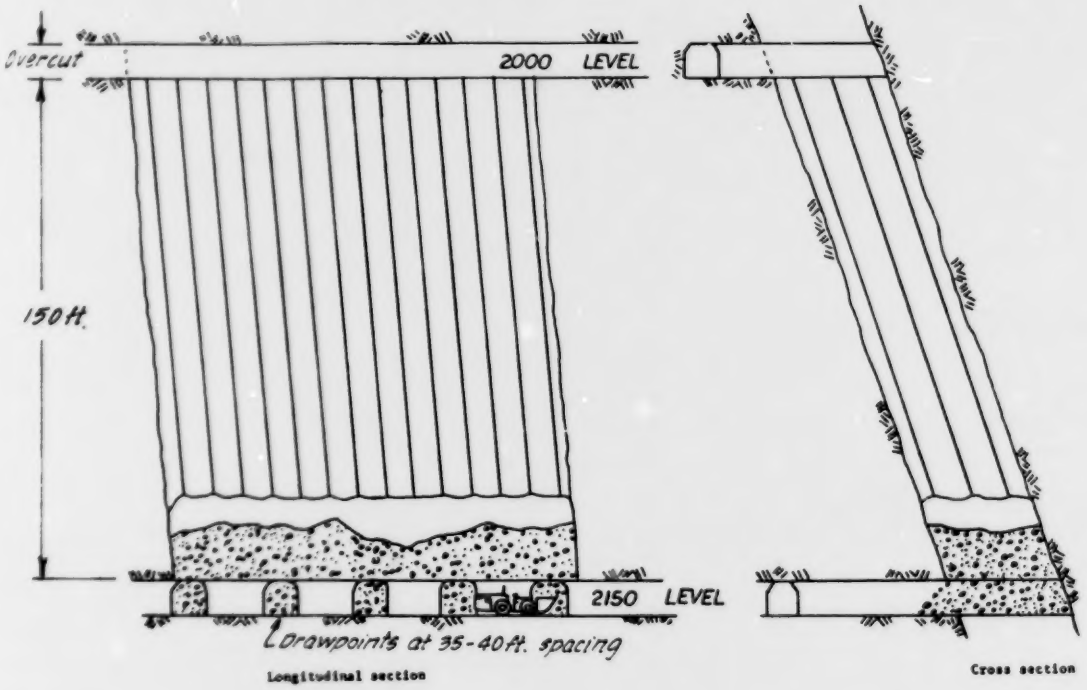
Figure 6-11

Reference: Golder, 1988

Golder Associates

VCR STOPING WITH PARALLEL HOLE DRILLING HOMESTAKE MINE

Figure **6-12**



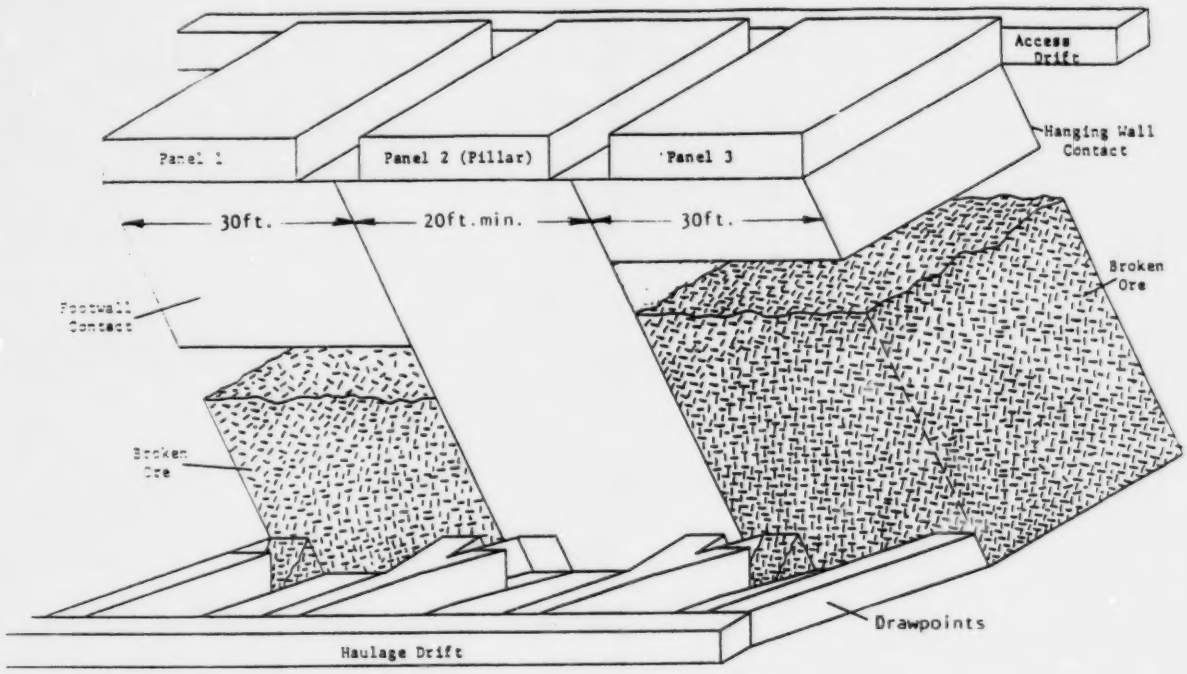
Not to Scale.

PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

Reference: Mitchell, 1980

VCR STOPING WITH TRANSVERSE PANELS IN WIDE OREBODIES

Figure **6-13**



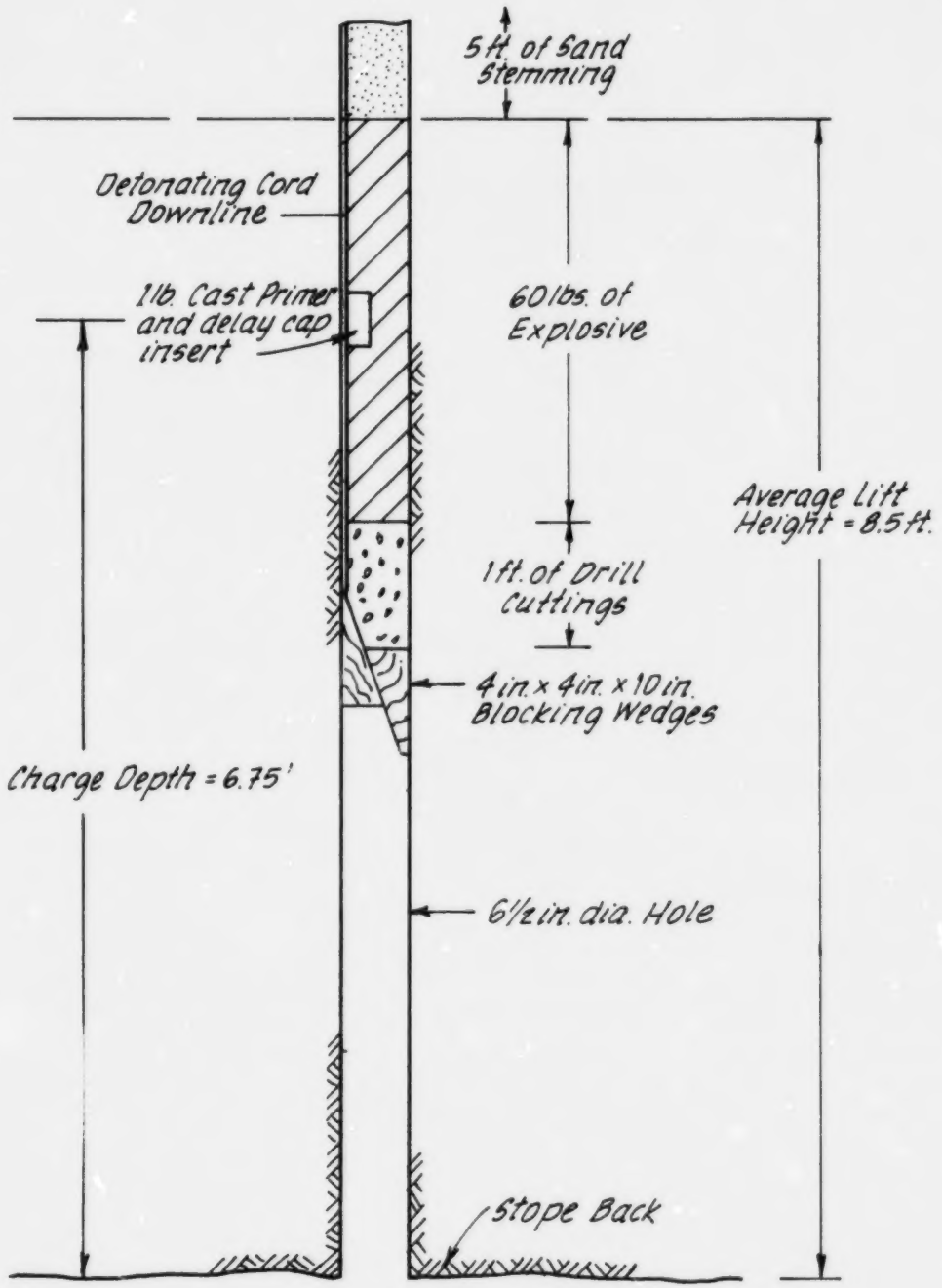
(NOT TO SCALE)

PROJECT NO..... DRAWN..... REVIEWED..... DATE.....

Reference : Orr, 1983

VCR CHARGE CONFIGURATION
HOMESTAKE MINE

Figure **6-14**


5ft. of Sand Stemming
Detonating Cord Downline
1lb. Cast Primer and delay cap insert
60lbs. of Explosive
Average Lift Height = 8.5ft.
1ft. of Drill Cuttings
4in. x 4in. x 10in. Blocking Wedges
Charge Depth = 6.75'
6½in. dia. Hole
Stope Back

Not to Scale.

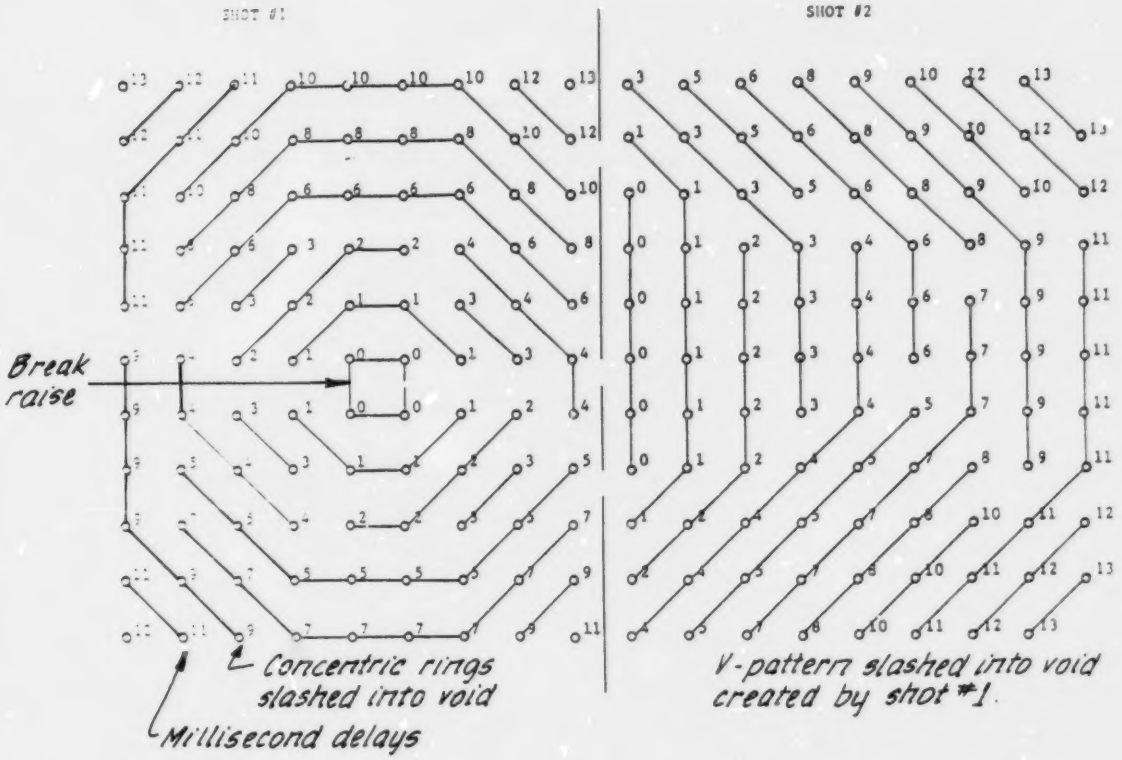
Reference: Orr, 1983

PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

TYPICAL TIMING PATTERN FOR A TWO PHASE VCR BLAST HOMESTAKE MINE

Figure **6-15**

64" Dia. Hole on an 8'x8' Pattern

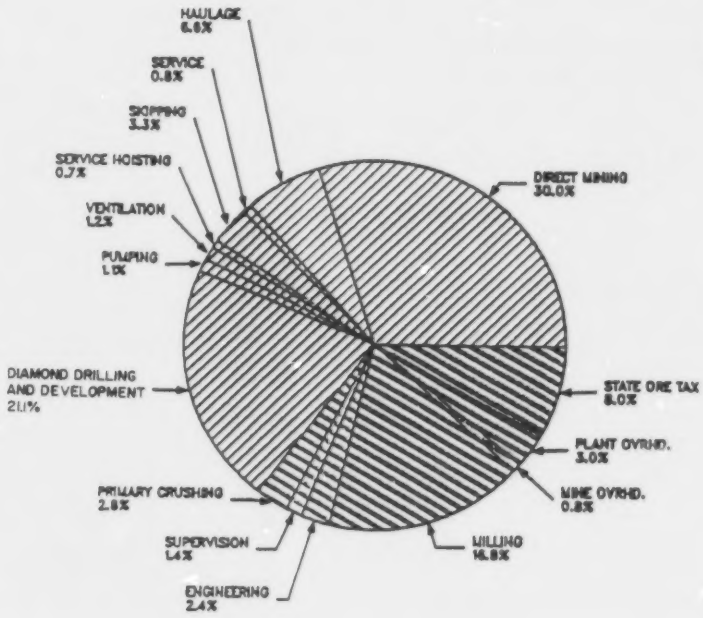


Reference : Orr, 1983

PROJECT NO. ..... DRAWN ..... REVIEWED ..... DATE .....

DETAILED STOPING COSTS FOR VCR MINING
HOMESTAKE MINE

Figure **6-16**

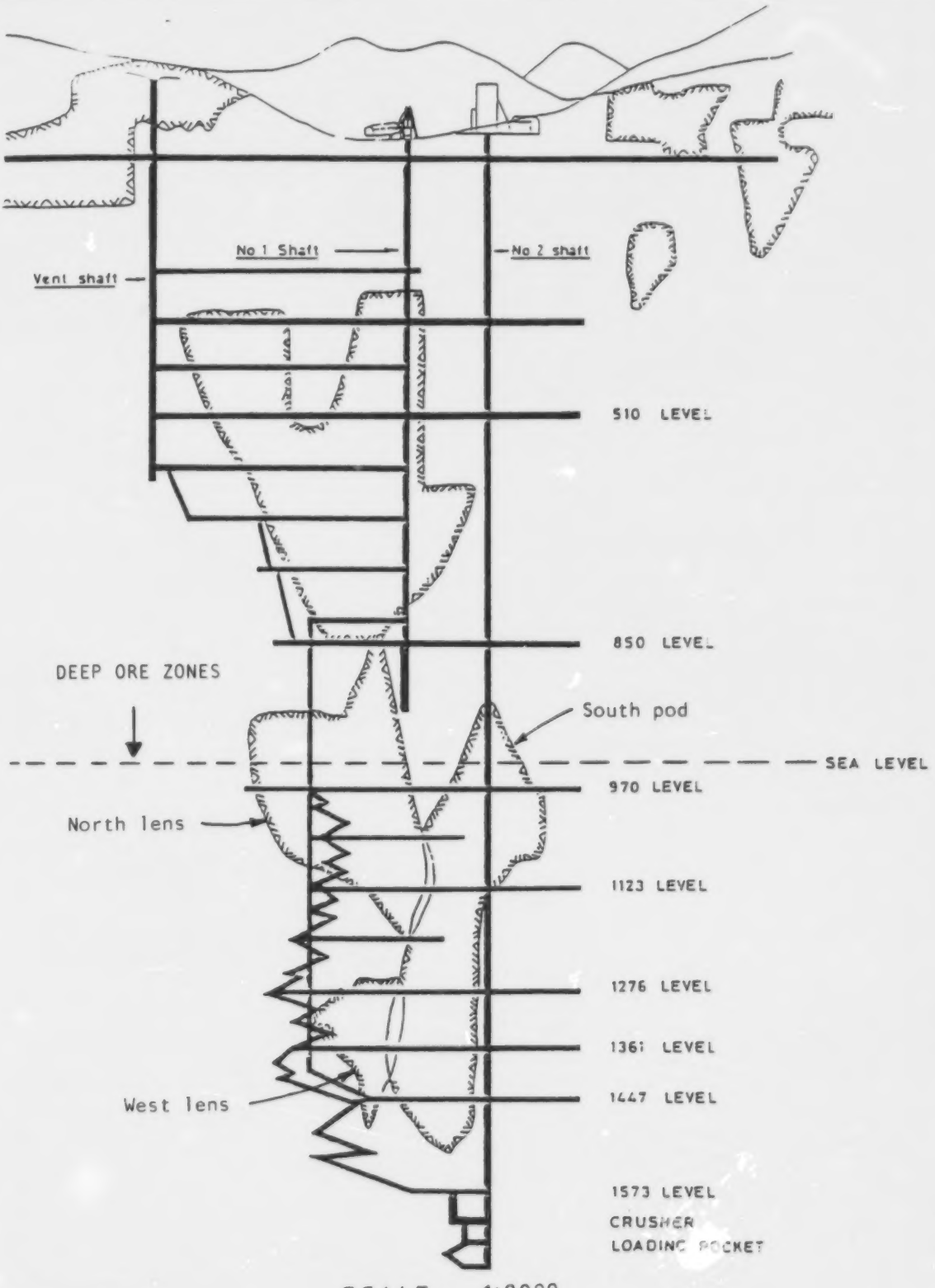


PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

Reference: Orr, 1983

CAROLUSBERG MINE
CROSS SECTION

Figure **6-17**



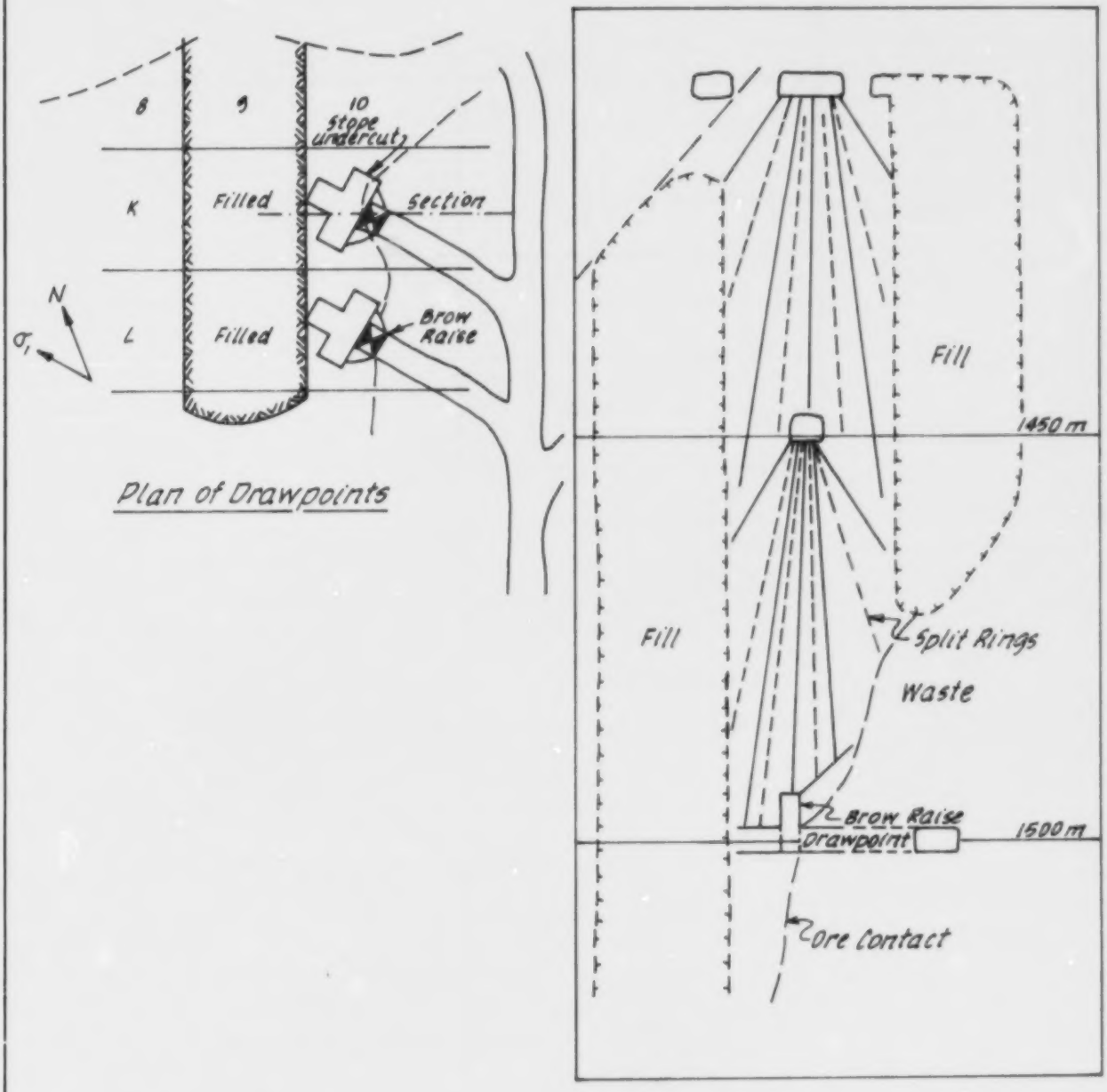
SCALE 1:8000

Reference: Langley-Cook, 1985

PROJECT NO. ..... DRAWN ..... REVIEWED ..... DATE .....

DRILLHOLE AND DRAWPOINT LAYOUT
CAROLUSBERG DEEP OREBODY

Figure **6-18**

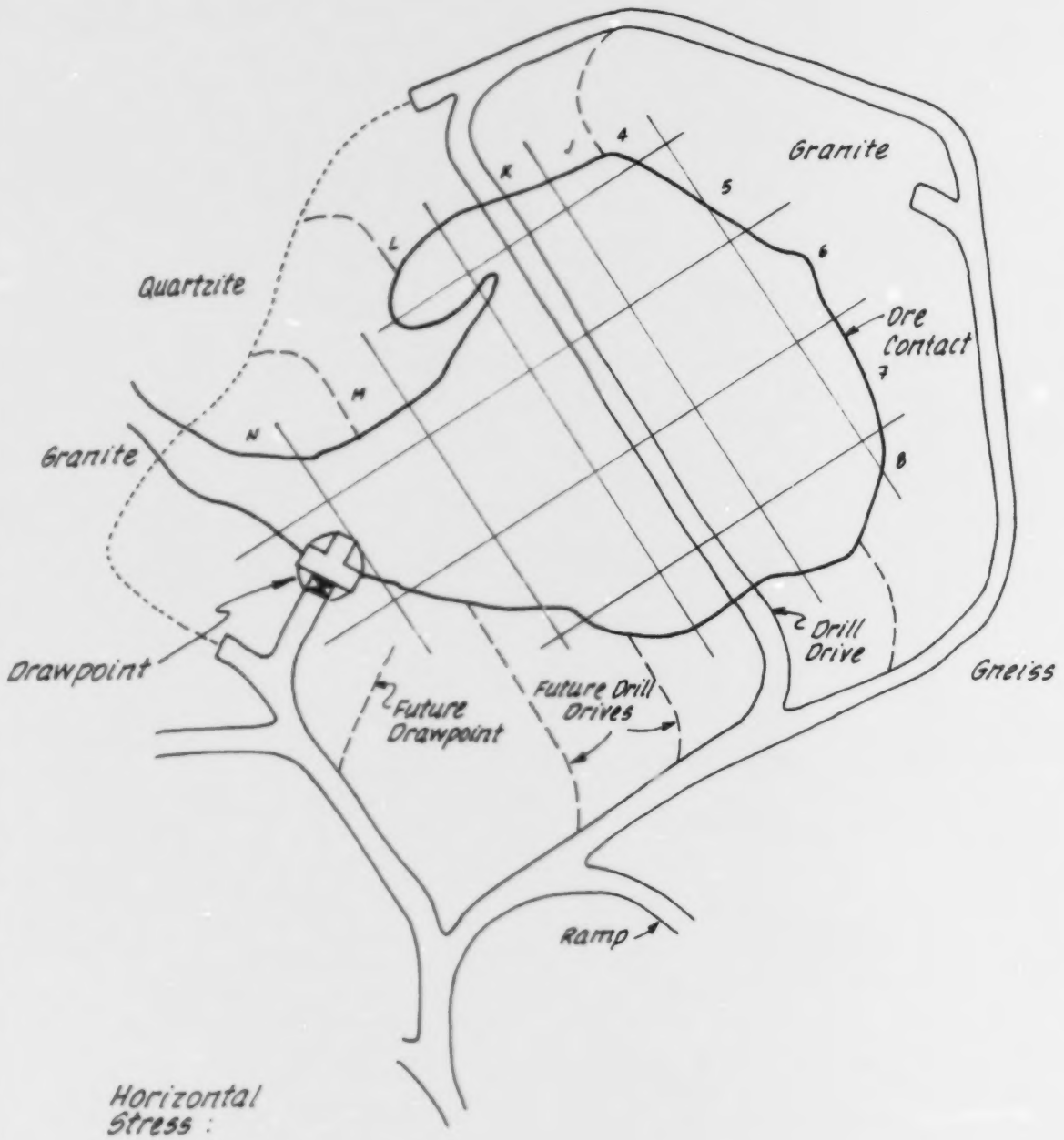


Reference: Bryant, 1986

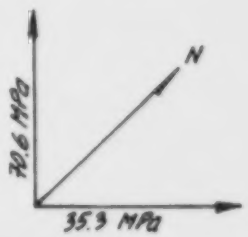
PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

STOPING PLAN - 1482 LEVEL
CAROLUSBERG DEEP OREBODY

Figure **6-19**



Horizontal Stress :



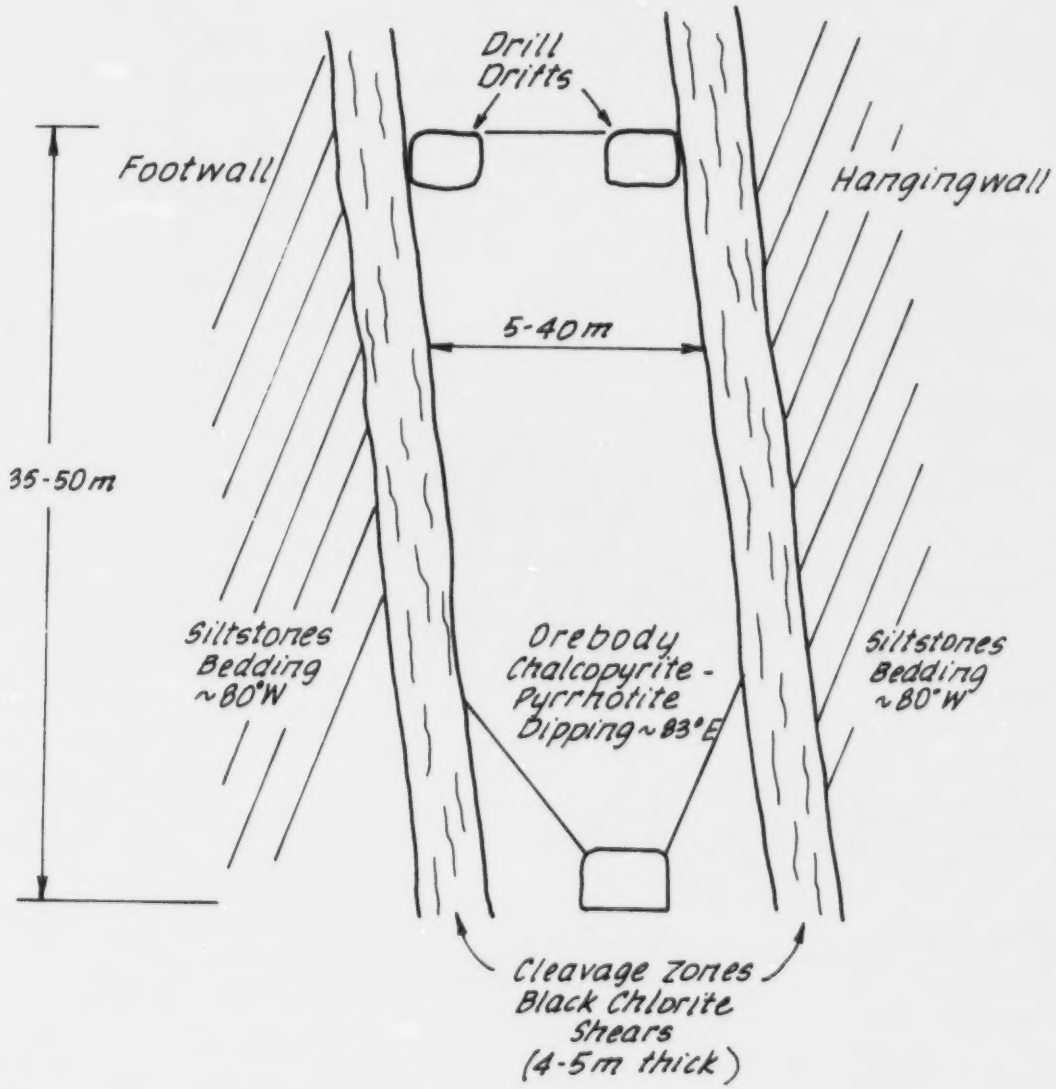
Scale 1:1000

Reference: Longley - Cook, 1985

PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

TYPICAL GEOLOGIC SECTION
CSA MINE

Figure **6-20**



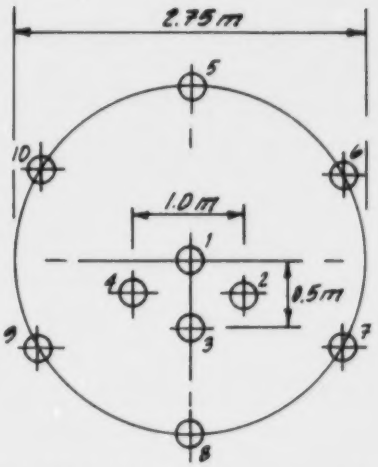
Not to Scale.

Reference: JKMRC, 1984

PROJECT NO. ..... DRAWN ..... REVIEWED ..... DATE .....

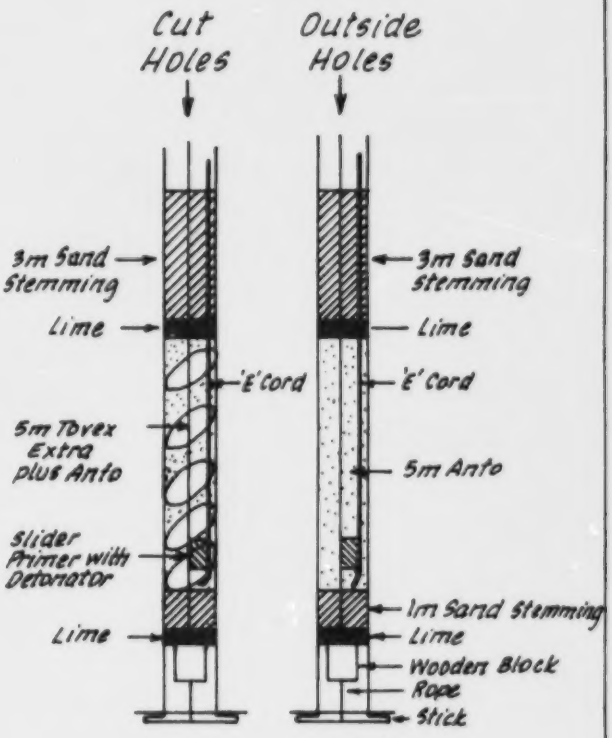
LONGHOLE RAISE DEVELOPMENT
CSA MINE

Figure **6-21**

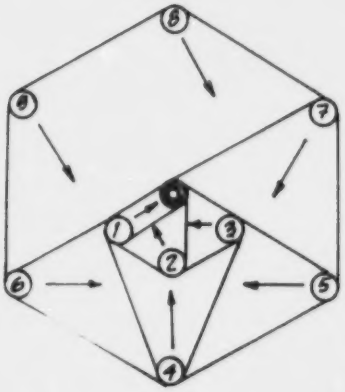


Note: All holes 165mm diameter.

Standard Raise Pattern



Method of Hole Charging



⑥ Hole with firing sequence
→ Direction of firing
Note: Middle hole uncharged

Idealized Delay Pattern

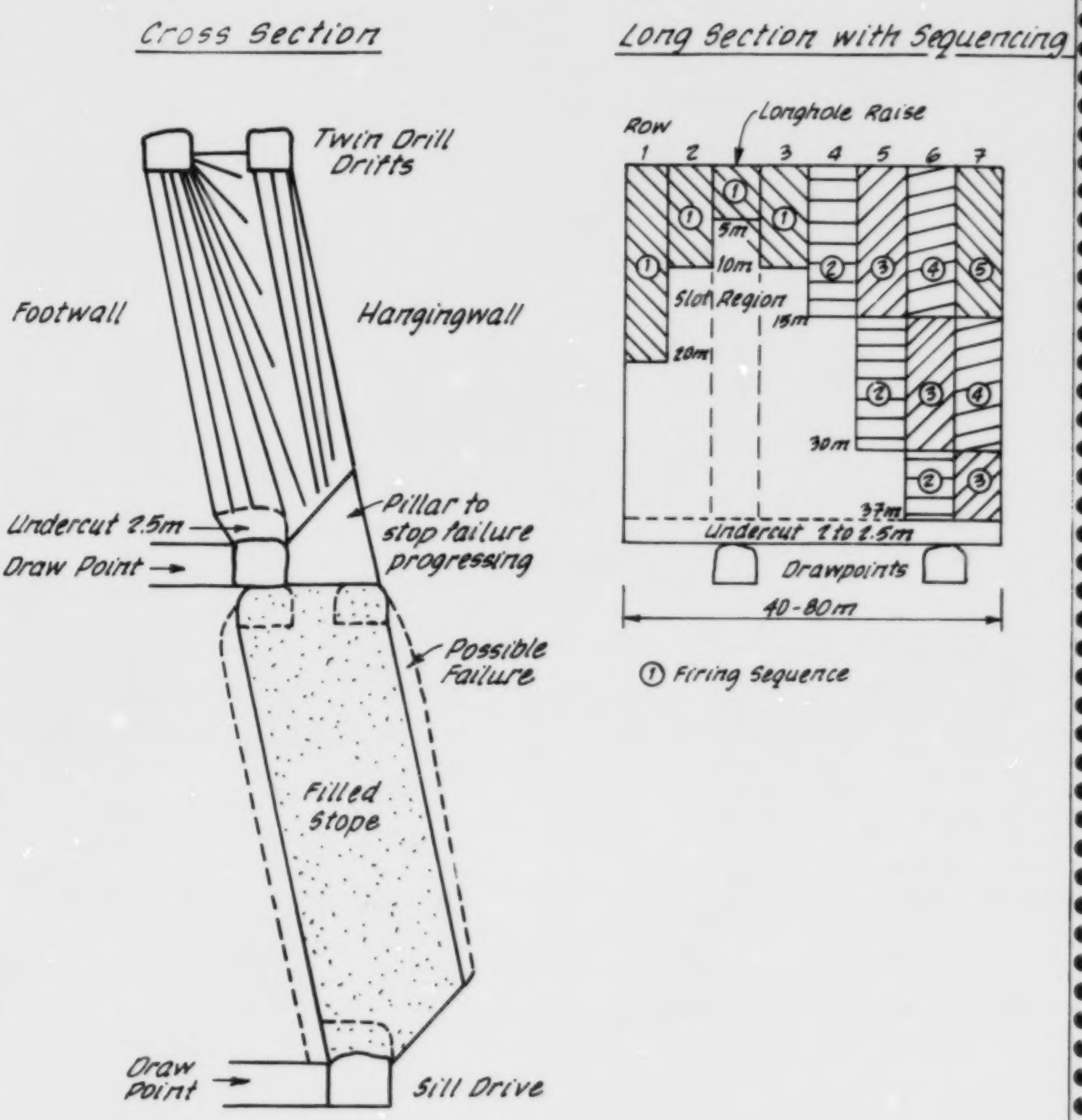
Reference: Price, 1983

Not to Scale.

PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

TYPICAL STOPE LAYOUT
CSA MINE

Figure **6-22**



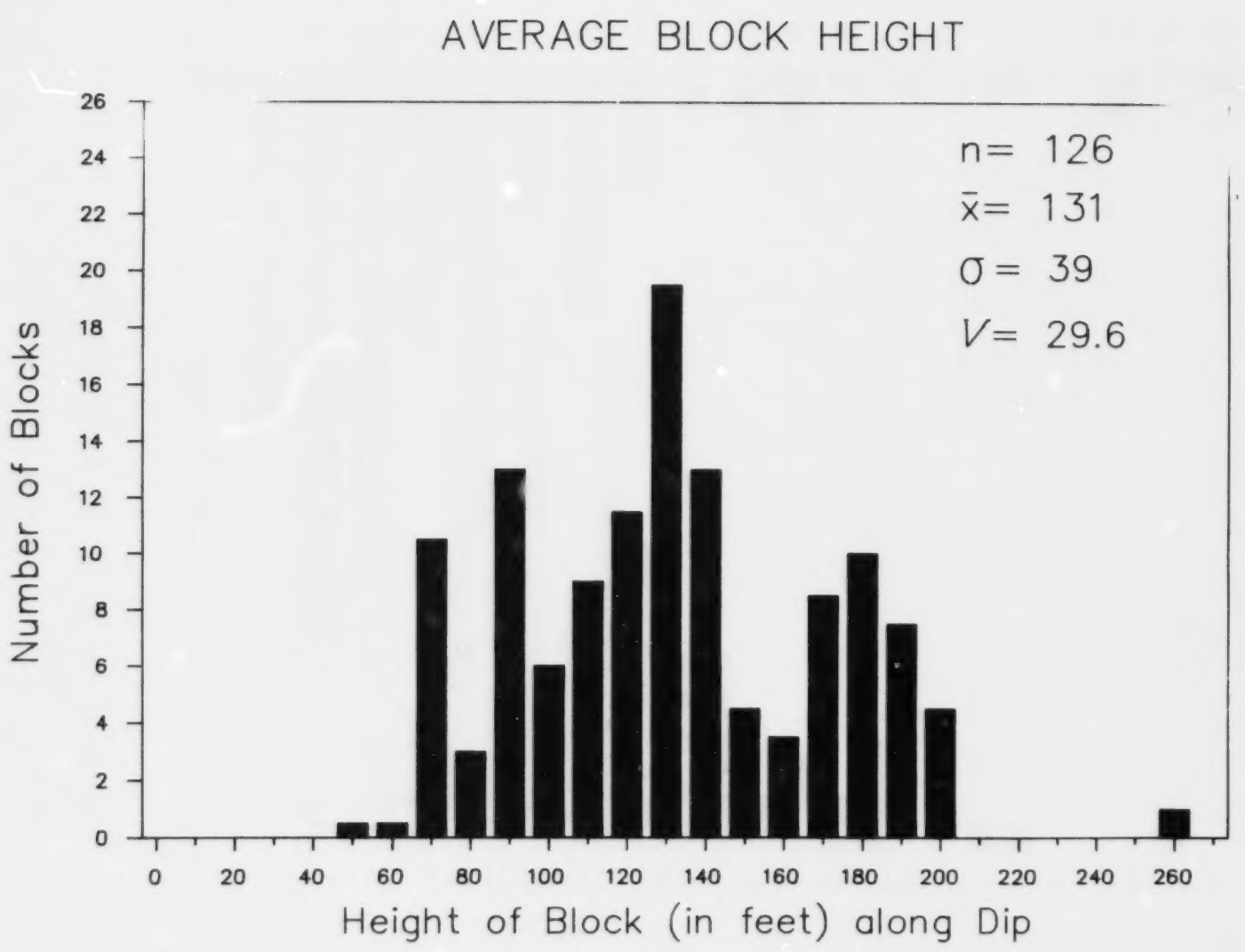
Not to Scale.

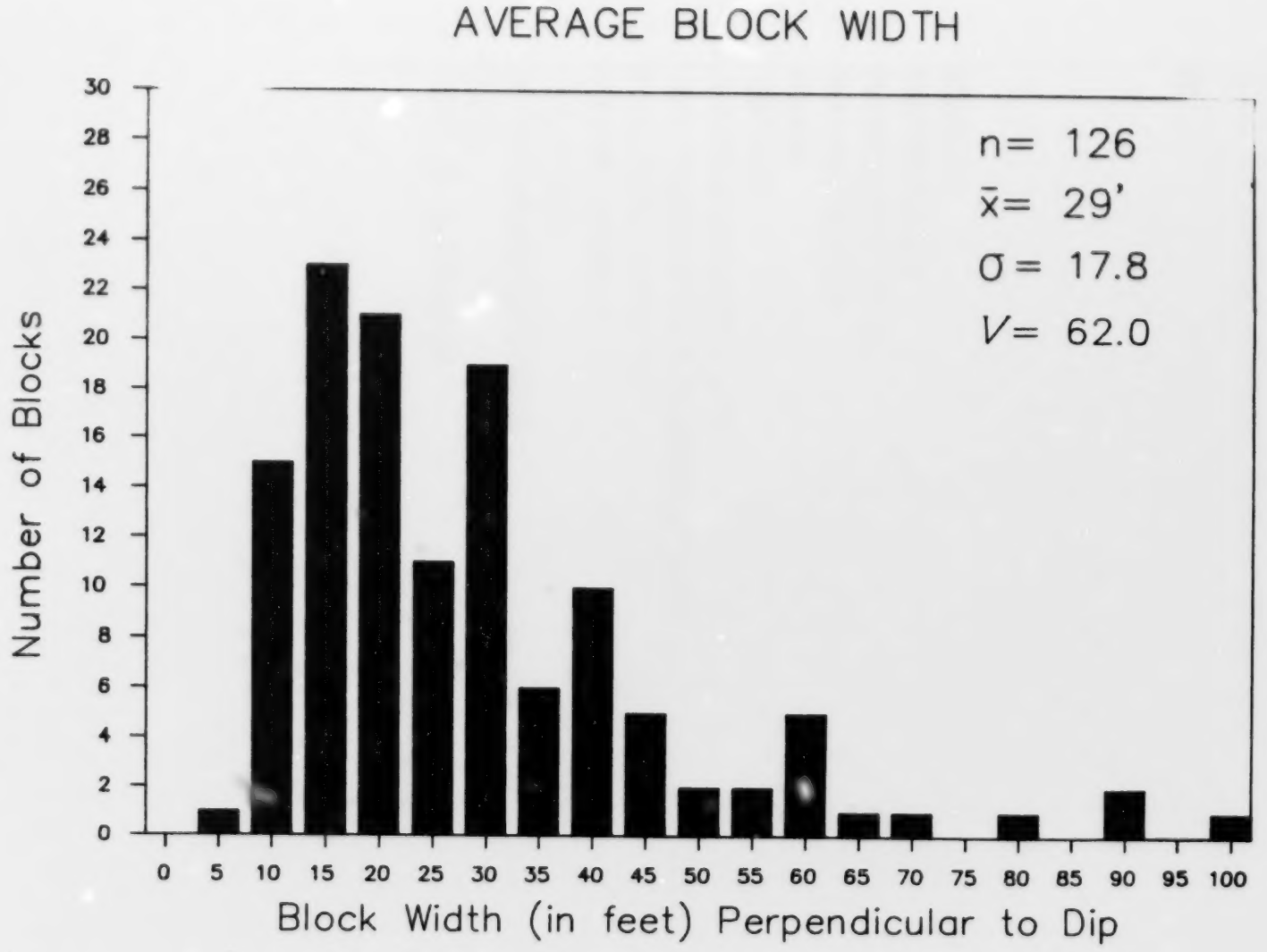
Reference: Munro, 1985

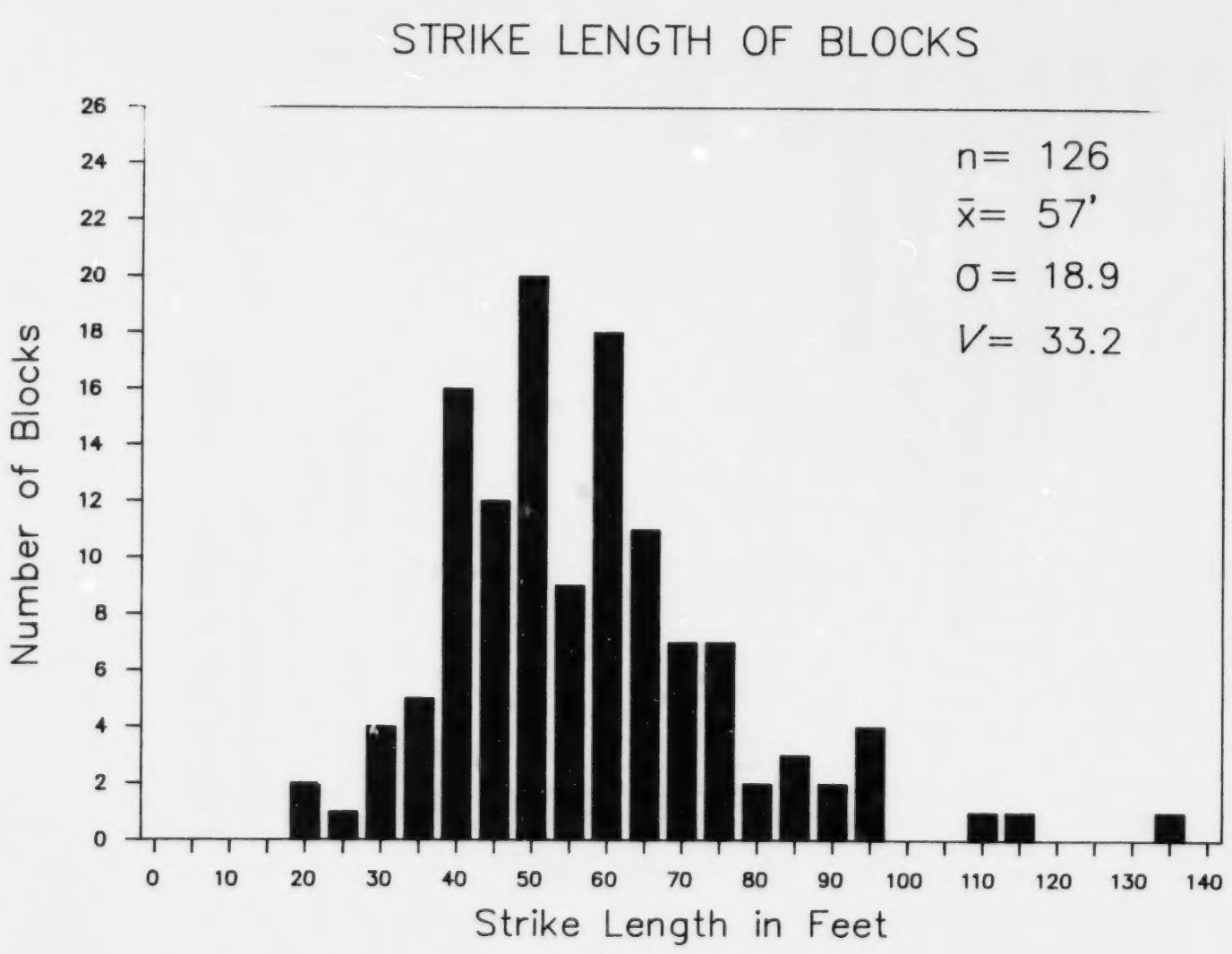
PROJECT NO DRAWN REVIEWED DATE

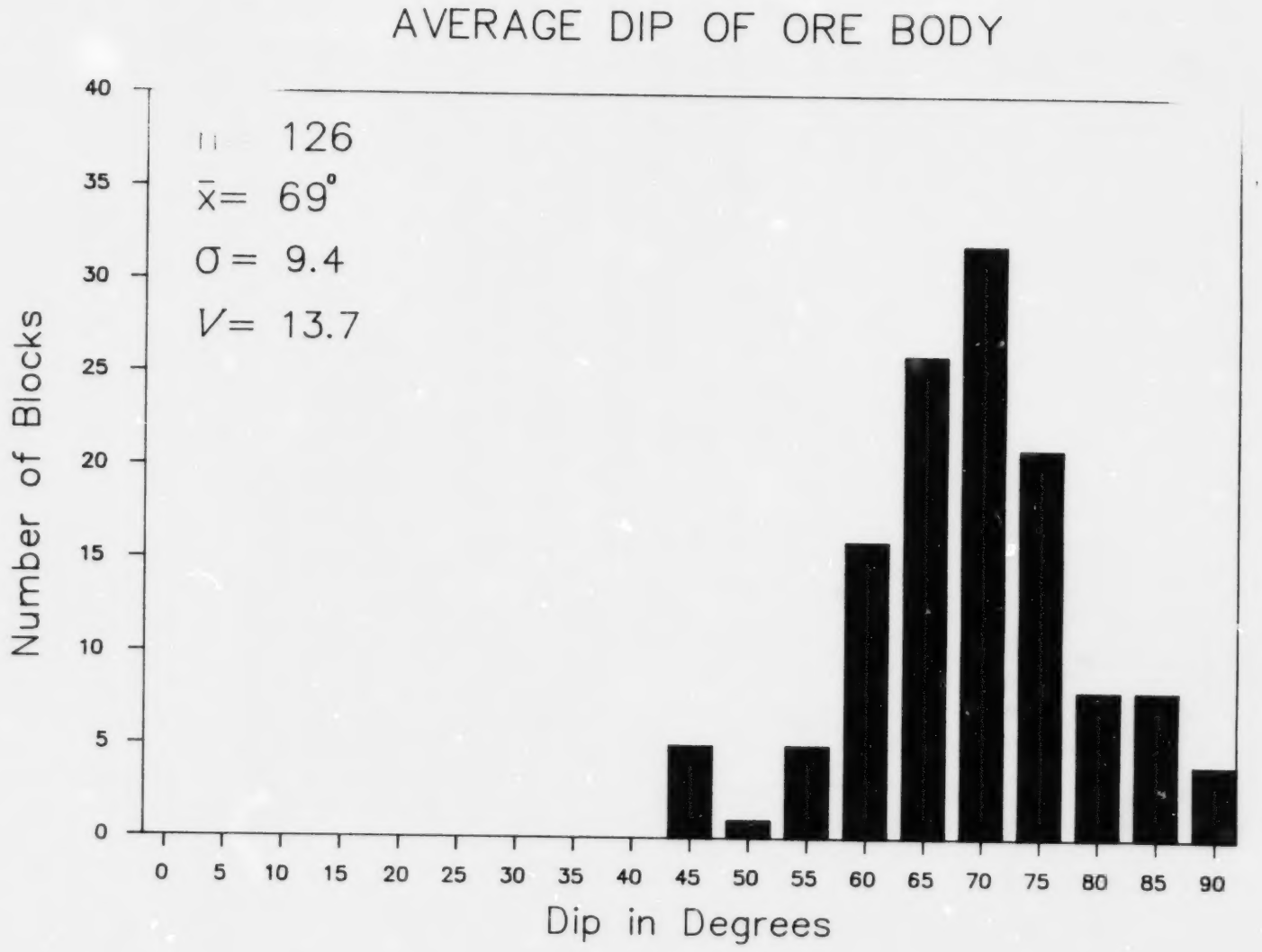
APPENDIX 6-A

HISTOGRAMS OF VBM BLOCK SIZES, OREBODY DIP AND POWDER FACTORS

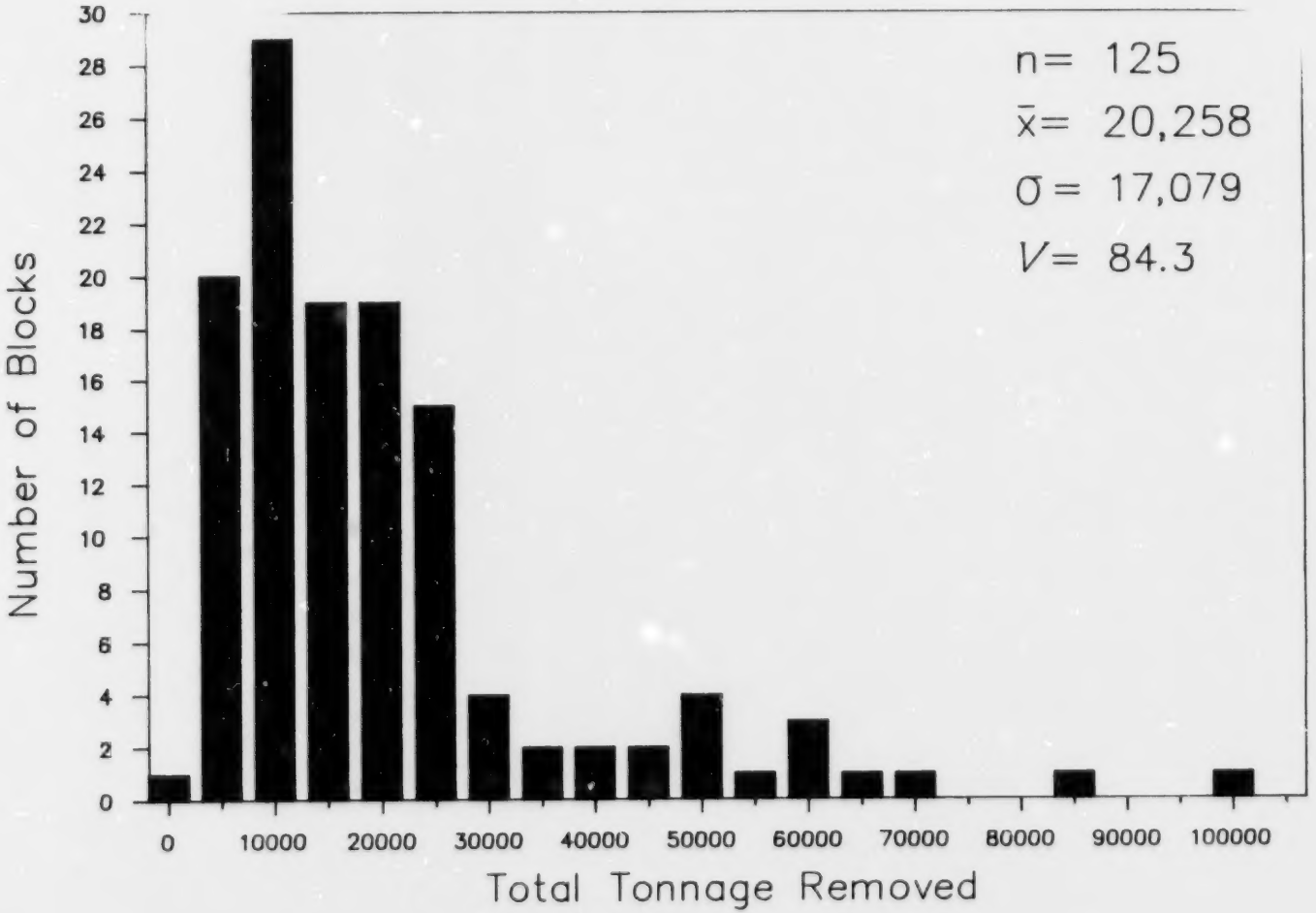

AVERAGE BLOCK HEIGHT
n= 126
x̄= 131
σ= 39
V= 29.6
Number of Blocks
0
2
4
6
8
10
12
14
16
18
20
22
24
26
0
20
40
60
80
100
120
140
160
180
200
220
240
260
Height of Block (in feet) along Dip

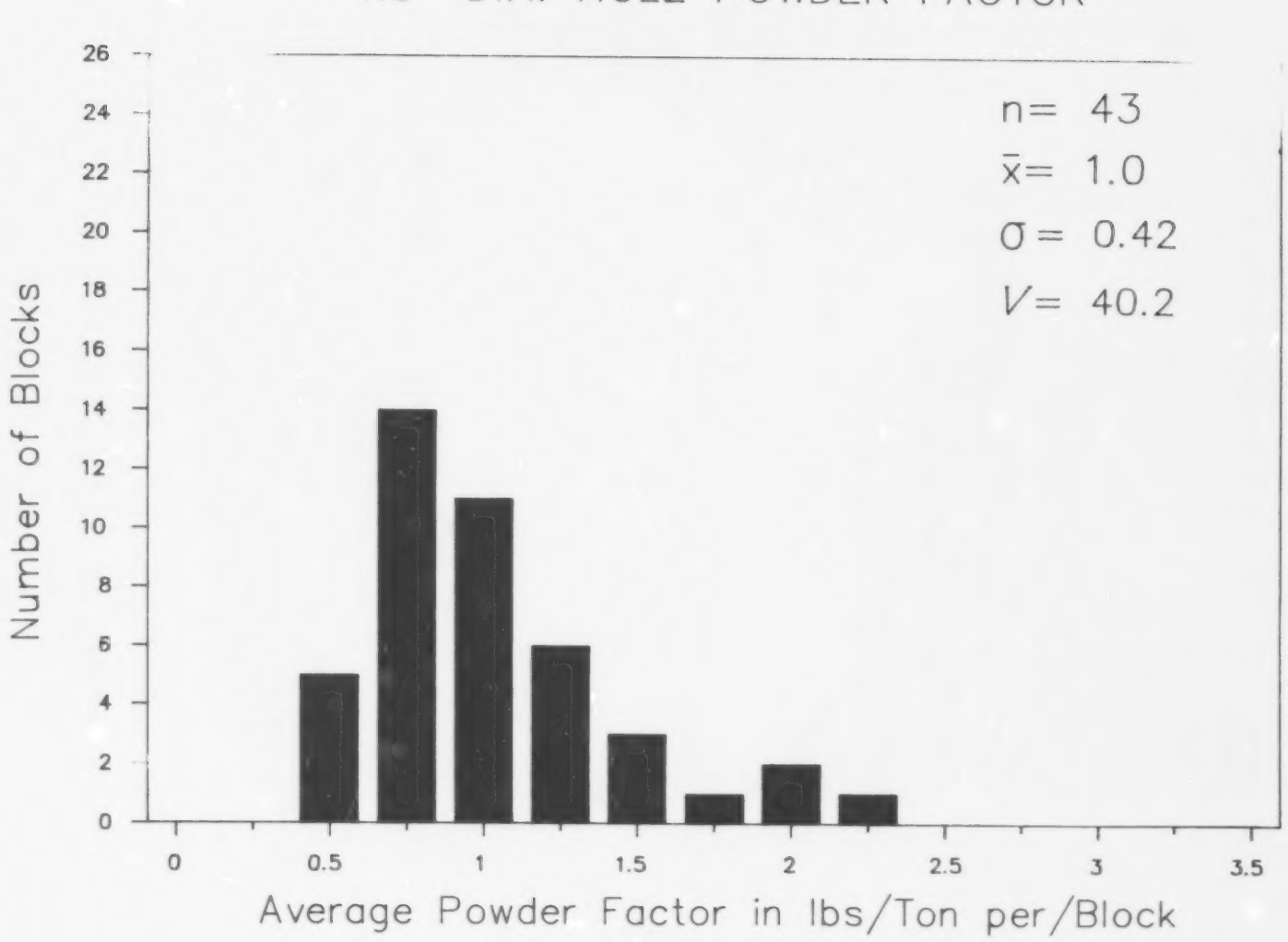

AVERAGE BLOCK WIDTH
n= 126
x̄= 29'
σ= 17.8
V= 62.0
Number of Blocks
0
2
4
6
8
10
12
14
16
18
20
22
24
26
28
30
0 5 10 15 20 25 30 35 40 45 50 55 60 65 70 75 80 85 90 95 100
Block Width (in feet) Perpendicular to Dip

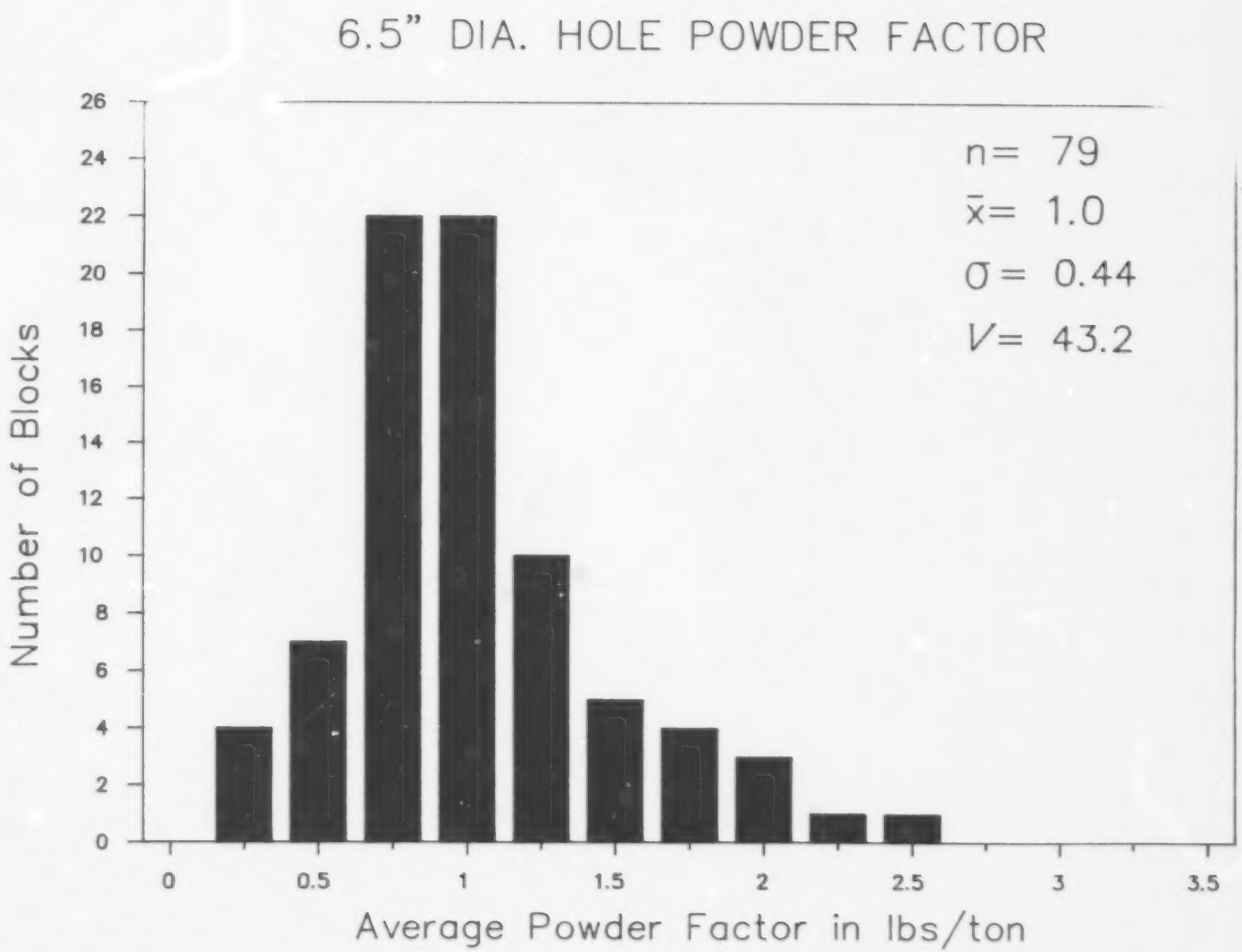

STRIKE LENGTH OF BLOCKS
n= 126
x̄= 57'
σ= 18.9
V= 33.2
Number of Blocks
0
2
4
6
8
10
12
14
16
18
20
22
24
26
0
10
20
30
40
50
60
70
80
90
100
110
120
130
140
Strike Length in Feet


AVERAGE DIP OF ORE BODY
n = 126
x̄ = 69°
σ = 9.4
V = 13.7
Number of Blocks
0
5
10
15
20
25
30
35
40
0 5 10 15 20 25 30 35 40 45 50 55 60 65 70 75 80 85 90
Dip in Degrees

BLOCK TONNAGE CALCULATED BY DRAWPOINT COUNT




4.5" DIA. HOLE POWDER FACTOR
n= 43
x̄= 1.0
σ= 0.42
V= 40.2
Number of Blocks
26
24
22
20
18
16
14
12
10
8
6
4
2
0
0
0.5
1
1.5
2
2.5
3
3.5
Average Powder Factor in lbs/Ton per/Block


6.5" DIA. HOLE POWDER FACTOR
n= 79
x̄= 1.0
σ= 0.44
V= 43.2
Number of Blocks
0
2
4
6
8
10
12
14
16
18
20
22
24
26
0
0.5
1
1.5
2
2.5
3
3.5
Average Powder Factor in lbs/ton

APPENDIX 6-B

MINING EQUIPMENT AT THOMPSON

MINING EQUIPMENT AT THOMPSON

Quantity	Description
2	Wagner ST2A scooptrams
26	Wagner ST2B scooptrams (6 remote-controlled)
6	Wagner ST2D scooptrams (1 remote-controlled)
1	Jarvis Clark JS220E scooptram
2	Jarvis Clark JS200 scooptrams (1 remote-controlled)
12	Jarvis Clark JS350 scooptrams (all remote-controlled)
5	Jarvis Clark JS350B scooptrams (all remote-controlled)
2	Jarvis Clark JS500 scooptrams (both remote-controlled)
4	Jarvis Clark JS600 scooptrams (all remote-controlled)
6	Eimco 911 scooptrams
2	Toyota utility vehicles
1	955 utility vehicle
1	975 scissor lift
1	D3 cat
1	D4 cat
1	Quad ATV vehicle
2	JDT 420 trucks
1	JDT 426B truck
1	545 Ford tractor
2	Jarvis Clark 2 boom pneumatic jumbo drill, rail mounted
1	Atlas Copco 2 boom pneumatic jumbo drill, tire mounted
6	Gardner Denver 2 boom hydraulic jumbo drill, tire mounted
2	Gardner Denver 1 boom hydraulic jumbo drill, tire mounted
1	MacLean 1 boom hydraulic rock bolter, tire mounted
1	Ingersoll Rand 4.5" 1TH drill
5	Mission 6.5" ITH drill
3	Mission 4.5" ITH drill, with compressor
1	GO60 4.5" ITH drill
7	CD360 6.5" ITH drill
2	CD360 4.5" ITH drill
1	CD360B 4.5" ITH drill, with compressor
16	Booster compressors
343	Jackleg drills; Seco-319 and Canun-24
144	Stoper drills; Seco-97, Joy-12, Canun-35
7	CP65 drills
1	VEG drill

10	S36 Secan longhole drills
7	Longhole drill carriers
3	Robbins 61R raisebore drills
4	Pettibone loaders
8	Diesel locomotives; 8, 9, 13 ton
42	Battery locomotives; 3.5, 8,9 ton
4	Trolley locomotives
29	Eimco air loaders
5	Eimco 21 backhoe/ditch cleaners
141	Tugger hoists
66	Slusher hoists
3	Shotcrete machines
95	Hudson cars; 56 cu. ft.
340	Granby cars; 110 cu. ft.
90	Ore cars; 140 cu. ft..
63	Timber cars
37	Fuel cars

APPENDIX 6-C

BACKFILL BARRICADE INSTALLATION PROCEDURE AT INCO THOMPSON MINE

BACKFILL BARRICADE CONSTRUCTION PROCEDURE
(for widths up to 16 ft.)

See Figure 6C-6 for the barricade construction drawing. Most VBM sandfill barricades are 16 ft. wide or less. The timber design of drawpoint and longitudinal retreat barricades are the same, although different methods of anchoring the bottom of the barricades have been discussed previously. Barricades of over 16 ft. in width and over 7 ft. high require approval from the Mine Engineer to ensure proper K-brace location and size.

Material Required:

- 8" flat x 11' spruce
- 10" x 12" x 16' fir timber
- 2" x 4", 2" x 6", 2" x 8" timber
- sausage rolls
- fabrene
- cement
- 2" diameter x 4' long dowel pins
- 5" x 10" nails
- 3/8" x 10" x 10" steel bearing plates

The VBM barricades are to be a minimum of 12 ft. away fromthe block brow. Locate the position of the barricade and blast one hitch for the K-brace in each wall of the drawpoint or stope. The hitch location is at 5' grade and is to be about 6 ft. from the barricade position on the extraction drift side of the drawpoint. K-braces must be angled between 40 and 50 degrees fromthe barricade.

Install all of the 8" flat spruce except for one or the centre uprights to allow access through the barricade. Stand the uprights so the full face of the barricade is covered and nail them to the stope side of the top and bottom whalers. Position all drain pipes (and the sandfill pipe if required) through the barricade timber and install the pressure gauge. See Figure 6C-7 for the pressure gauge assembly.

Drill the holes for the 2" diameter dowel pins at about 1 ft. spacing along the top and bottom whalers. There should be one pin for every foot of barricade width along the top and barricade width along the top and bottom whalers. Install the 2" x 4' dowel pins and wedge the whalers in place.

Position the double 10" x 12" x 16' fir centre whalers to the walls. If the barricade is 12 ft. wide or less, the whalers can be of 10" x 10" fir timber. Drill two 2'-6" holes in each wall at the ends of the centre whaler for 2" diameter dowel pins. Install 3/8" 10" x 10" steel bearing plates on the end of the centre whaler and install the dowel pins.

Cut the remaining 8" flat spruce uprights at the centre of the middle whaler and nail the top section of the upright to the top and middle whalers. Build a door for the bottom portionof the barricade without the 8" flat uprights. See Figure 6C-8 for the door drawing. The barricade door may be located above the centre whaler when the barricade is built above the old fill.

Sheet the barricade on the stope side with 2" planks. Cut the timber around the perimeter of the barricade to fit the rock contour as closely as possible. Seal the perimeter of the barricade with a burlap sausage roll and/or cement seal.

Cover the barricade with double fabrene, but do not install fabrene over the barricade door. Allow 4' of extra slack at theend of the fabrene for a skirt along the drift walls, floor and back. Leave about 6" of slack in the fabrene around the barricade perimeter toallow for slight barricade movement.

Install the door in the barricade and nail it to the 8" uprights on each side of the door. Place the remaining half of the cut 8" flat spruce uprights in position and nail to the bottom (or top) and centre whalers.

Figure 6C-9 shows how to cut the K-braces to obtain the proper angles. Install the 10" x 12" fir K-braces to the centre whaler and nail it in place. If the barricade is 12' wide or less, the K-brace and whalers can be made of 10" x 10" fir timber. The K-brace must fit solidly into the wall hitches. If the wall hitches are not shaped properly, install a 3/8" x 10" x 10" steel bearing plate on the ends of the K-brace and wedge the bearing plate to the wall. The K-brace should be between 40 and 50 degrees to the wall.

VBM barricade K-branches should be anchored in blasted wall hitches wherever possible. If poor ground conditions prevent the blasting of a proper wall hitch, the K-brace(s) may be anchored by pinning the end of the K-brace to the wall. See Figure 6C-10 for the pinned K-brace standard. note that a timber strap will have to be installed to hold the K-brace in place.

Vertical K-braces or posting the barricade must have Mine Engineering approval before construction.

All VBM barricades are to built in accordance with standard timbering practice. Wedges can be used to fill gaps and block timber, but too many wedges will weaken the structure and the barricade will fail inspection. Under no conditions are wedges allowed at the K-brace joint between the the K-brace and centre whaler.

Drain Towers

Material Required:

- 8' wire screen
- 2" x 4" timber
- 2" x 6" timber
- burlap socks

plastic drain pipe

Construct six 2" x 6" cross braces 18" long. Nail the cross braces to 4 lengths of 8' long 2" x 4" timber to form a 8' long drain tower frame. The cross braces are to be located at 2 ft. centers.

Roll one length of 8' screen around the wooden frame and nail the screen to the 2" x 4" timbers.

A minimum of two drain towers are required for each barricade. The bottom elevation of each drain tower is to be about 18" above the floor elevation to allow for proper drainage.

If two drain towers are used, each drain toweer is to hve its own 4" diameter plastic drain pipe through the barricade. If four drain towers are used, two of the drain towers may be jointed with 4" plastic drain pipe in the stope as long as there are two 4" diameter drain pipes coming through the barricade. Connect the drain pipes tightly to the burlap sock to avoid leaks. The drain pipes are to go through the barricade about 6" above the top of the bottom whalers. See Figure 6C-11 for drain tower construction. If a slimes catch is constructed in front of the barricade, the two drain pipes must be brought through the slimes catch to prevent sand from blocking off the drain pipes. Plastic drain pipes may be replaced with steel pipes if desired.

Perforated Drain Pipe

Perforated drain pipes are required only in VBM blocks with solid walls and bench where all water must drain through the barricade. A minimum of three perforated drain pipes will be installed. The drain pipes are to be brought through the top of any available barricade(s) in the block.

Each drain pipe must have a burlap sock installed around the outside of the pipe to prevent sand from entering the pipe. The 50 ft. long socks must have an overlap of at least 1 ft. The socks must be tied to the drain pipe at 3 ft. intervals.

The drain pipes are installed from the top sill and are brought out through the block drawpoint(s). The top of the drain pipes are to be sealed and fastened to the screen on the back of the top sill.

Fir VBM Barricades

In some cases a VBM barricade may have to drain water over a number of years as VBM blocks above are mined and filled. In such cases, the spruce uprights on the bottom (and oldest) barricades may rot. Use 10" x 10" x 11' fir uprights placed on 1 ft. centers when building barricades that will remain in service for a long period of time. See Figure 6C-6 for the fir barricade drawing.

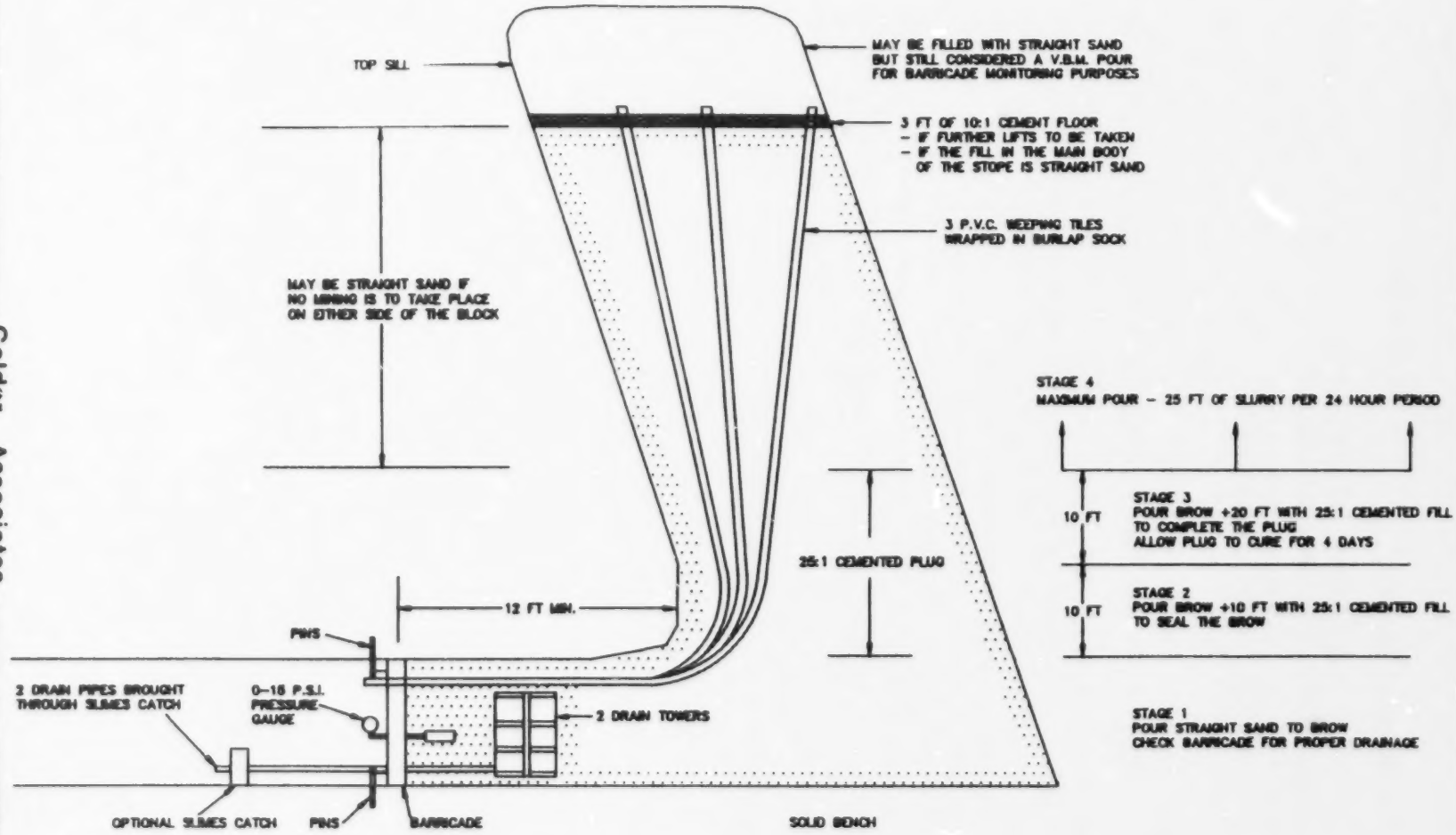
Barricade Construction Report

Upon completion of barricade construction and the drainage system, an inspection is to be completed by the General Foreman or his designate. It is the responsibility of the General Foreman to verify that the barricade and drainage system are fabricated to standard practice. His report will be handed to the Safety Officer and relayed to the Mines Inspector.

Reference: Inco Ltd., 1988, "Thompson Mine Standard Practice VBM Power Procedure".

PROJECT NO. 852-1605 DRAWN — REVIEWED DATE Nov. 1989

VBM POUR GENERAL ARRANGEMENT BLOCK WITH SOLID BENCH AND WALLS

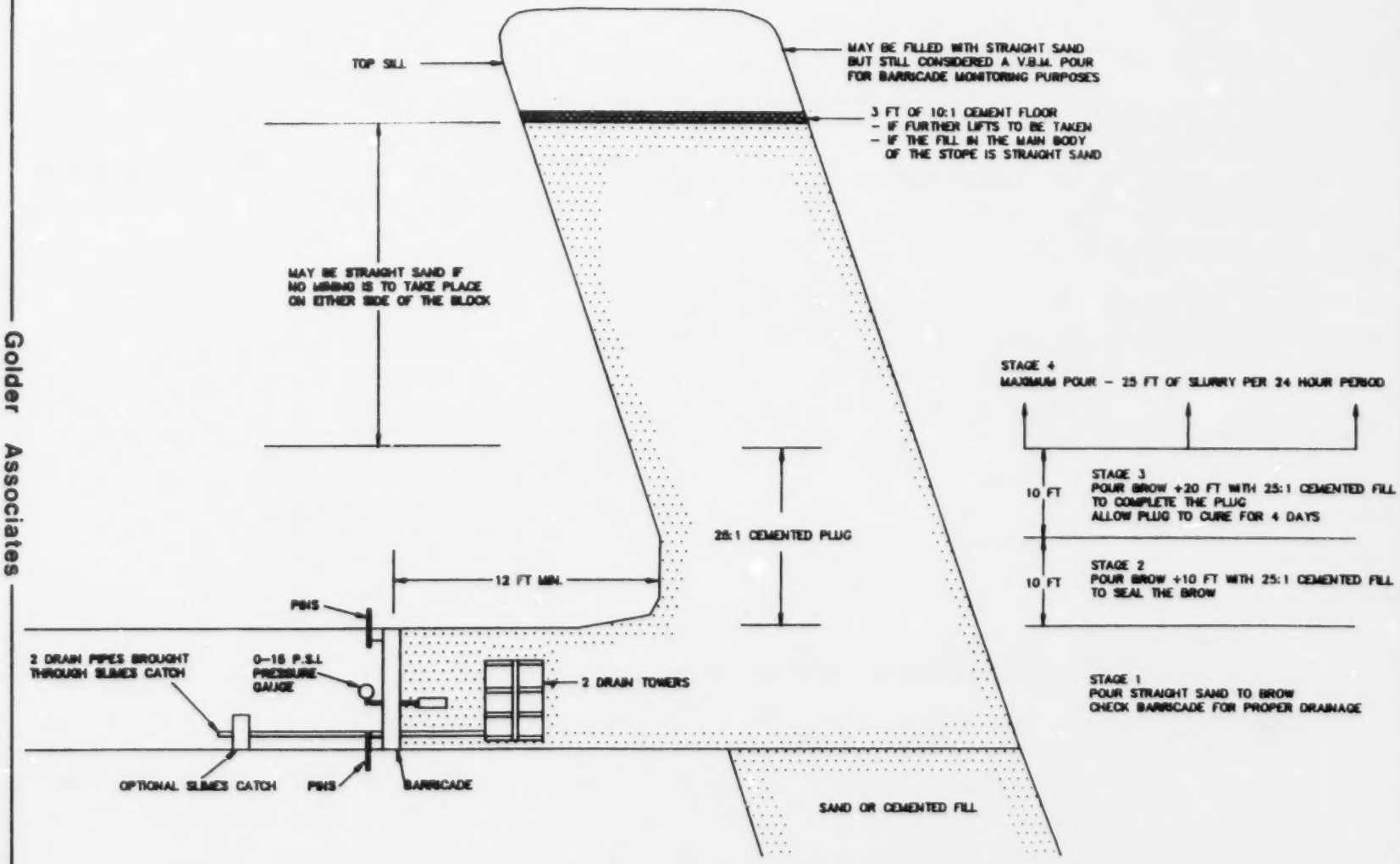

TOP SILL
MAY BE FILLED WITH STRAIGHT SAND BUT STILL CONSIDERED A V.B.M. POUR FOR BARRICADE MONITORING PURPOSES
3 FT OF 10:1 CEMENT FLOOR
- IF FURTHER LIFTS TO BE TAKEN
- IF THE FILL IN THE MAIN BODY OF THE STOPE IS STRAIGHT SAND
3 P.V.C. WEEPING TILES WRAPPED IN BURLAP SOCK
MAY BE STRAIGHT SAND IF NO MINING IS TO TAKE PLACE ON EITHER SIDE OF THE BLOCK
25:1 CEMENTED PLUG
12 FT MIN.
PINS
2 DRAIN PIPES BROUGHT THROUGH SLIMES CATCH
0-15 P.S.I. PRESSURE GAUGE
2 DRAIN TOWERS
OPTIONAL SLIMES CATCH
PINS
BARRICADE
SOLID BENCH
STAGE 4
MAXIMUM POUR - 25 FT OF SLURRY PER 24 HOUR PERIOD
10 FT
STAGE 3
POUR BROW +20 FT WITH 25:1 CEMENTED FILL TO COMPLETE THE PLUG
ALLOW PLUG TO CURE FOR 4 DAYS
10 FT
STAGE 2
POUR BROW +10 FT WITH 25:1 CEMENTED FILL TO SEAL THE BROW
STAGE 1
POUR STRAIGHT SAND TO BROW
CHECK BARRICADE FOR PROPER DRAINAGE

Reference : Inco, 1988

Golder Associates

Figure 6C-1

PROJECT NO. ..... DRAWN ..... REVIEWED ..... DATE .....


TOP SILL
MAY BE FILLED WITH STRAIGHT SAND BUT STILL CONSIDERED A V.B.M. POUR FOR BARRICADE MONITORING PURPOSES
3 FT OF 10:1 CEMENT FLOOR
- IF FURTHER LIFTS TO BE TAKEN
- IF THE FILL IN THE MAIN BODY OF THE STOPE IS STRAIGHT SAND
MAY BE STRAIGHT SAND IF NO MINING IS TO TAKE PLACE ON EITHER SIDE OF THE BLOCK
25:1 CEMENTED PLUG
12 FT MIN.
PINS
2 DRAIN PIPES BROUGHT THROUGH SLIMES CATCH
0-15 P.S.I. PRESSURE GAUGE
2 DRAIN TOWERS
OPTIONAL SLIMES CATCH
PINS
BARRICADE
SAND OR CEMENTED FILL
STAGE 4
MAXIMUM POUR - 25 FT OF SLURRY PER 24 HOUR PERIOD
10 FT
STAGE 3
POUR BROW +20 FT WITH 25:1 CEMENTED FILL TO COMPLETE THE PLUG
ALLOW PLUG TO CURE FOR 4 DAYS
10 FT
STAGE 2
POUR BROW +10 FT WITH 25:1 CEMENTED FILL TO SEAL THE BROW
STAGE 1
POUR STRAIGHT SAND TO BROW
CHECK BARRICADE FOR PROPER DRAINAGE

Golder Associates

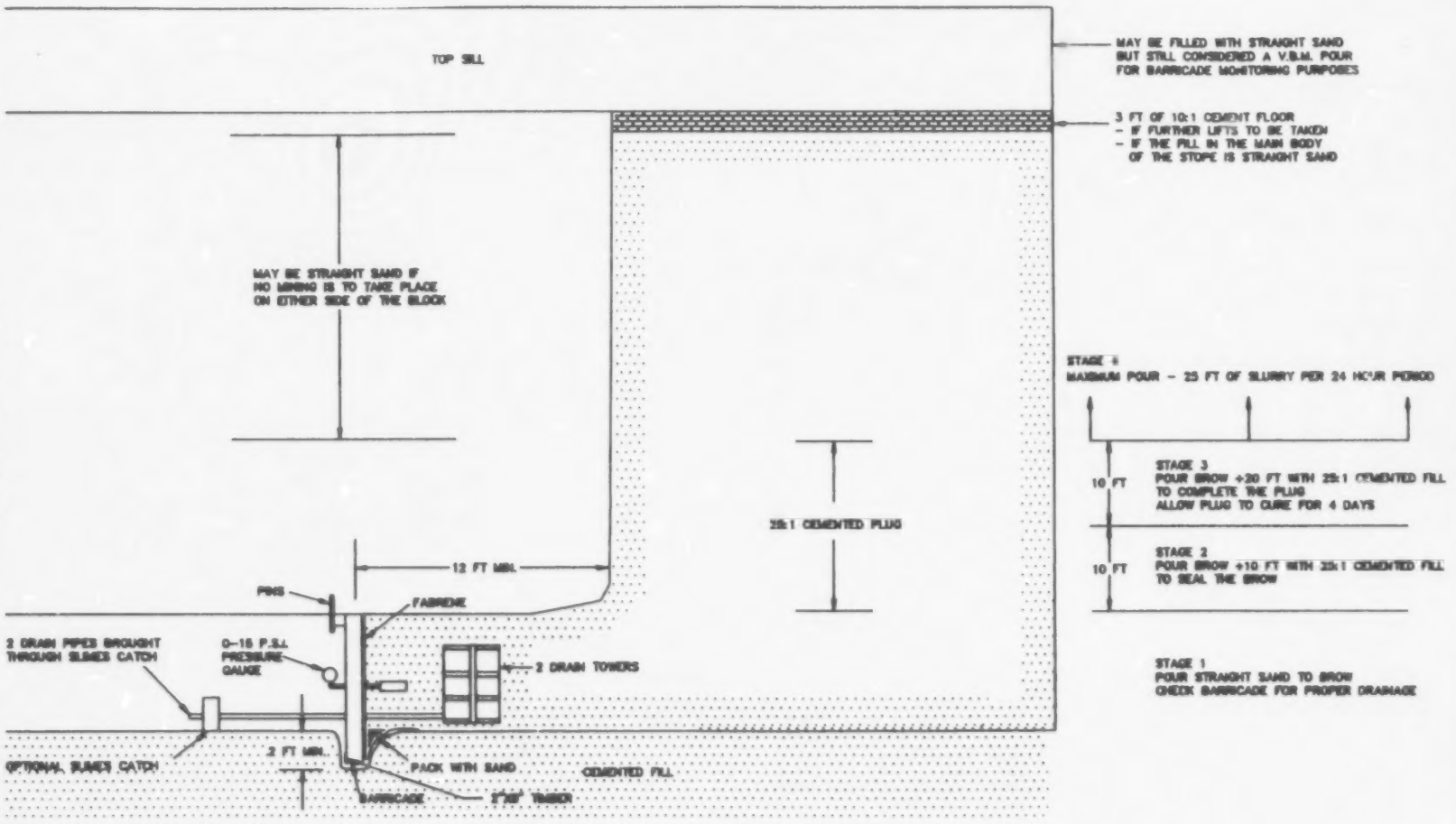
VBM POUR GENERAL ARRANGEMENT
DRAWPOINT BARRICADE, SAND AROUND BLOCK

Figure **6C-2**

Reference: Ivan 1984

PROJECT NO. 852-1605 DRAWN — REVIEWED DATE NOV. 1989

VBM POUR GENERAL ARRANGEMENT
LONGITUDINAL RETREAT
BARRICADE DUG INTO CEMENTED FILL


TOP SILL
MAY BE FILLED WITH STRAIGHT SAND BUT STILL CONSIDERED A V.B.M. POUR FOR BARRICADE MONITORING PURPOSES
3 FT OF 10:1 CEMENT FLOOR
- IF FURTHER LIFTS TO BE TAKEN
- IF THE FILL IN THE MAIN BODY OF THE STOPE IS STRAIGHT SAND
MAY BE STRAIGHT SAND IF NO MINING IS TO TAKE PLACE ON EITHER SIDE OF THE BLOCK
25:1 CEMENTED PLUG
STAGE 4
MAXIMUM POUR - 25 FT OF SLURRY PER 24 HOUR PERIOD
10 FT
STAGE 3
POUR BROW +20 FT WITH 25:1 CEMENTED FILL TO COMPLETE THE PLUG
ALLOW PLUG TO CURE FOR 4 DAYS
10 FT
STAGE 2
POUR BROW +10 FT WITH 25:1 CEMENTED FILL TO SEAL THE BROW
STAGE 1
POUR STRAIGHT SAND TO BROW
CHECK BARRICADE FOR PROPER DRAINAGE
12 FT MIN.
PINS
FABRENE
2 DRAIN PIPES BROUGHT THROUGH SLIMES CATCH
0-15 P.S.I. PRESSURE GAUGE
2 DRAIN TOWERS
OPTIONAL SLIMES CATCH
2 FT MIN.
PACK WITH SAND
CEMENTED FILL
BARRICADE
2"x8" TIMBER

Reference : Inco, 1988

Golder Associates

Figure 6C-3

PROJECT NO. 852-1605 DRAWN - REVIEWED DATE Nov. 189

VBM POUR GENERAL ARRANGEMENT
LONGITUDINAL RETREAT
BARRICADE DUG INTO UNCEMENTED FILL

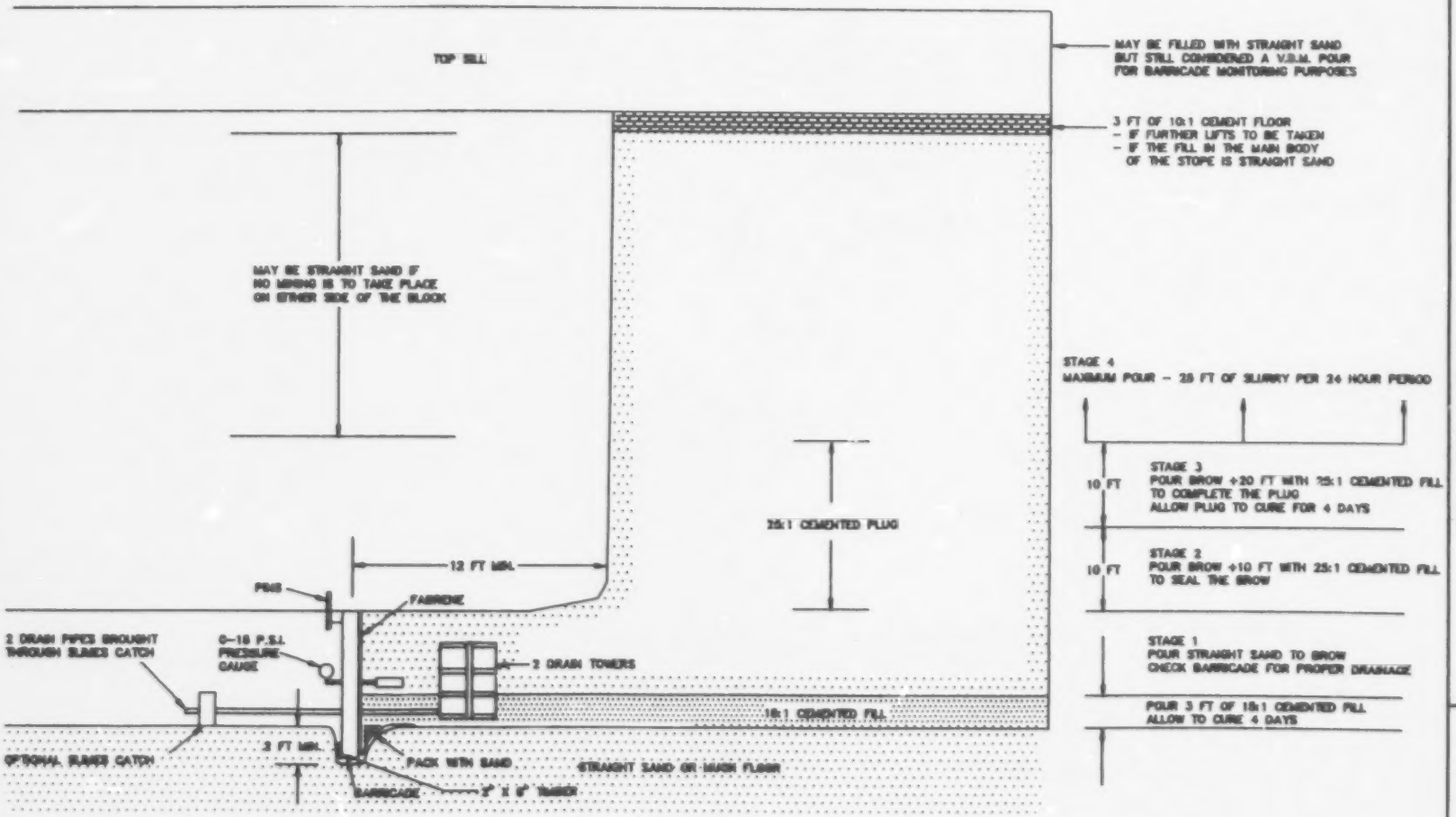
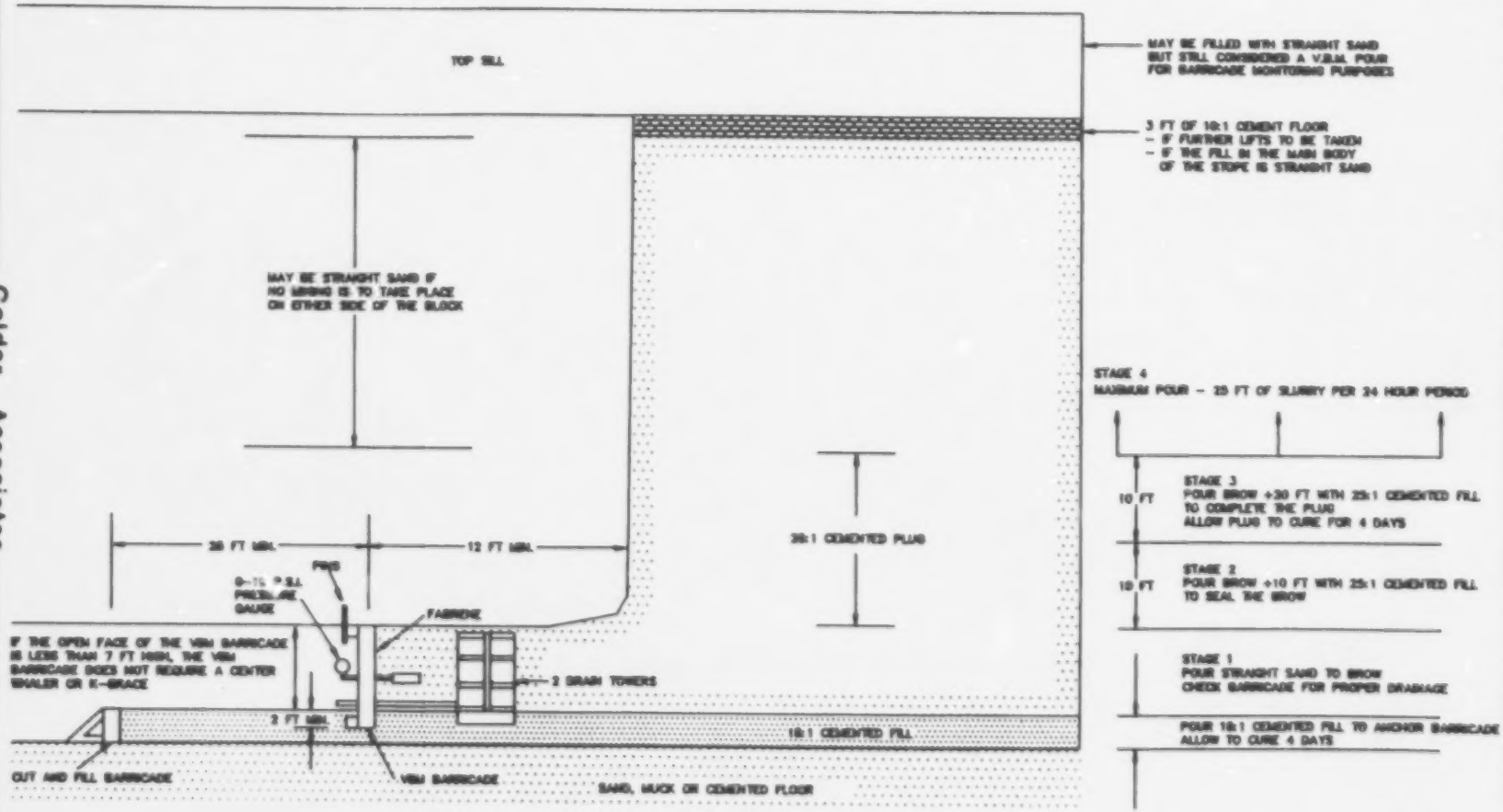

TOP SILL
MAY BE FILLED WITH STRAIGHT SAND BUT STILL CONSIDERED A V.B.M. POUR FOR BARRICADE MONITORING PURPOSES
3 FT OF 10:1 CEMENT FLOOR
- IF FURTHER LIFTS TO BE TAKEN
- IF THE FILL IN THE MAIN BODY OF THE STOPE IS STRAIGHT SAND
MAY BE STRAIGHT SAND IF NO MINING IS TO TAKE PLACE ON EITHER SIDE OF THE BLOCK
25:1 CEMENTED PLUG
12 FT MIN.
PEG
FABRIDIC
2 DRAIN PIPES BROUGHT THROUGH SLIMES CATCH
0-15 P.S.I. PRESSURE GAUGE
2 DRAIN TOWERS
10:1 CEMENTED FILL
OPTIONAL SLIMES CATCH
3 FT MIN.
PACK WITH SAND
STRAIGHT SAND OR MAIN FLOOR
BARRICADE
3" X 8" TIMBER
STAGE 4
MAXIMUM POUR - 25 FT OF SLURRY PER 24 HOUR PERIOD
10 FT
STAGE 3
POUR BROW +20 FT WITH 25:1 CEMENTED FILL TO COMPLETE THE PLUG
ALLOW PLUG TO CURE FOR 4 DAYS
10 FT
STAGE 2
POUR BROW +10 FT WITH 25:1 CEMENTED FILL TO SEAL THE BROW
STAGE 1
POUR STRAIGHT SAND TO BROW
CHECK BARRICADE FOR PROPER DRAINAGE
POUR 3 FT OF 10:1 CEMENTED FILL
ALLOW TO CURE 4 DAYS

Figure 6C-4

Golder Associates

Reference: INCO, 1988

PROJECT NO 852-1605 DRAWN — REVIEWED DATE Nov. 1989


TOP SILL
MAY BE FILLED WITH STRAIGHT SAND BUT STILL CONSIDERED A V.B.M. POUR FOR BARRICADE MONITORING PURPOSES
3 FT OF 10:1 CEMENT FLOOR
- IF FURTHER LIFTS TO BE TAKEN
- IF THE FILL IN THE MAIN BODY OF THE STOPE IS STRAIGHT SAND
MAY BE STRAIGHT SAND IF NO MINING IS TO TAKE PLACE ON EITHER SIDE OF THE BLOCK
20:1 CEMENTED PLUG
25 FT MIN.
12 FT MIN.
PIPE
0-15 P.S.I. PRESSURE GAUGE
FABRIDAM
IF THE OPEN FACE OF THE VBM BARRICADE IS LESS THAN 7 FT HIGH, THE VBM BARRICADE DOES NOT REQUIRE A CENTER WHALER OR K-BRACE
2 DRAIN TOWERS
2 FT MIN.
10:1 CEMENTED FILL
CUT AND FILL BARRICADE
VBM BARRICADE
SAND, MUCK OR CEMENTED FLOOR
STAGE 4
MAXIMUM POUR - 25 FT OF SLURRY PER 24 HOUR PERIOD
10 FT
STAGE 3
POUR BROW +20 FT WITH 25:1 CEMENTED FILL TO COMPLETE THE PLUG
ALLOW PLUG TO CURE FOR 4 DAYS
10 FT
STAGE 2
POUR BROW +10 FT WITH 25:1 CEMENTED FILL TO SEAL THE BROW
STAGE 1
POUR STRAIGHT SAND TO BROW
CHECK BARRICADE FOR PROPER DRAINAGE
POUR 10:1 CEMENTED FILL TO ANCHOR BARRICADE
ALLOW TO CURE 4 DAYS

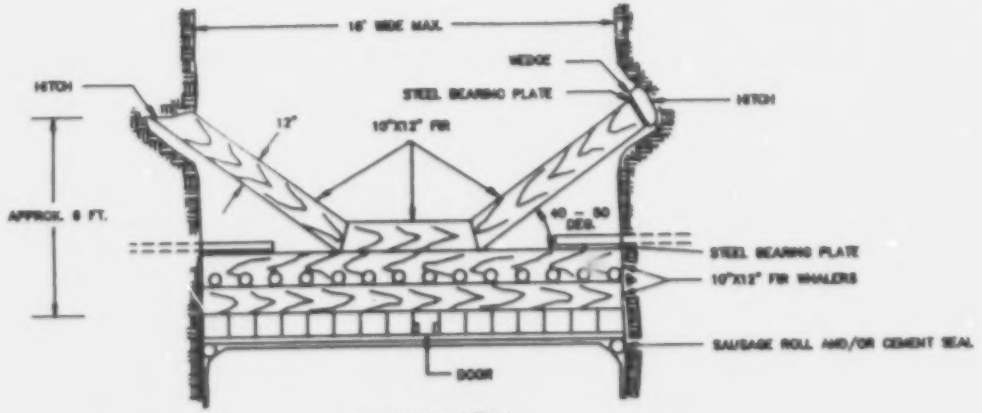
VBM POUR GENERAL ARRANGEMENT
LONGITUDINAL RETREAT
BARRICADE BUILT ABOVE OLD FILL FLOOR

Figure 6C-5

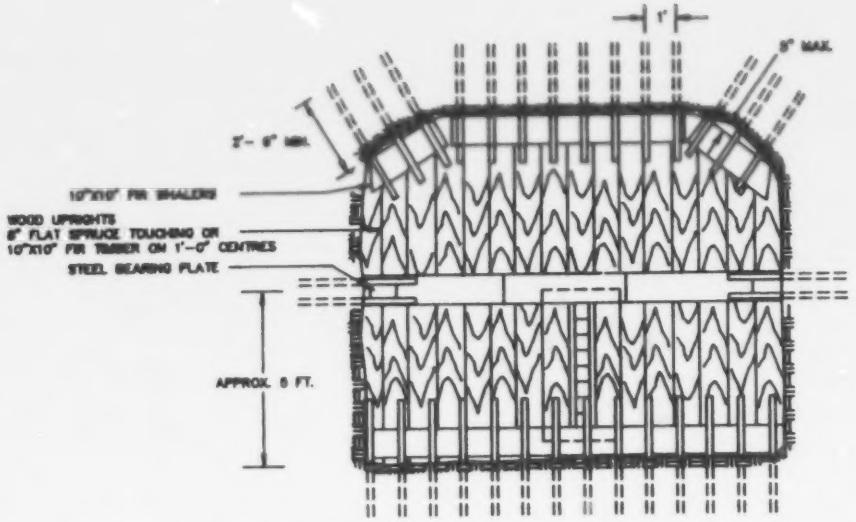
Golder Associates

Reference: Inco, 1988

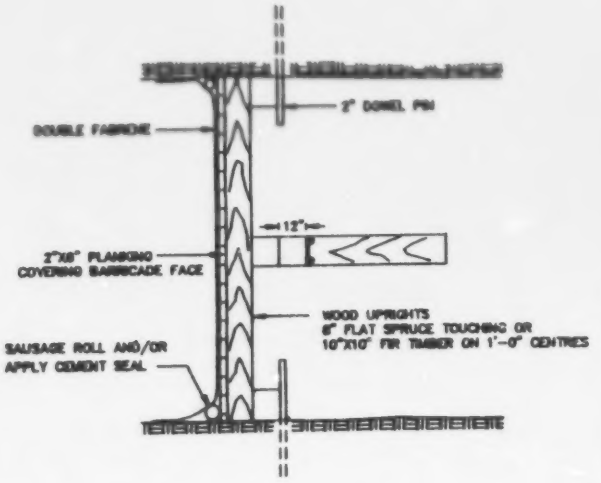
PROJECT NO. 852-1605 DRAWN — REVIEWED ..... DATE Nov. '89



PLAN VIEW



FRONT VIEW



FRONT VIEW

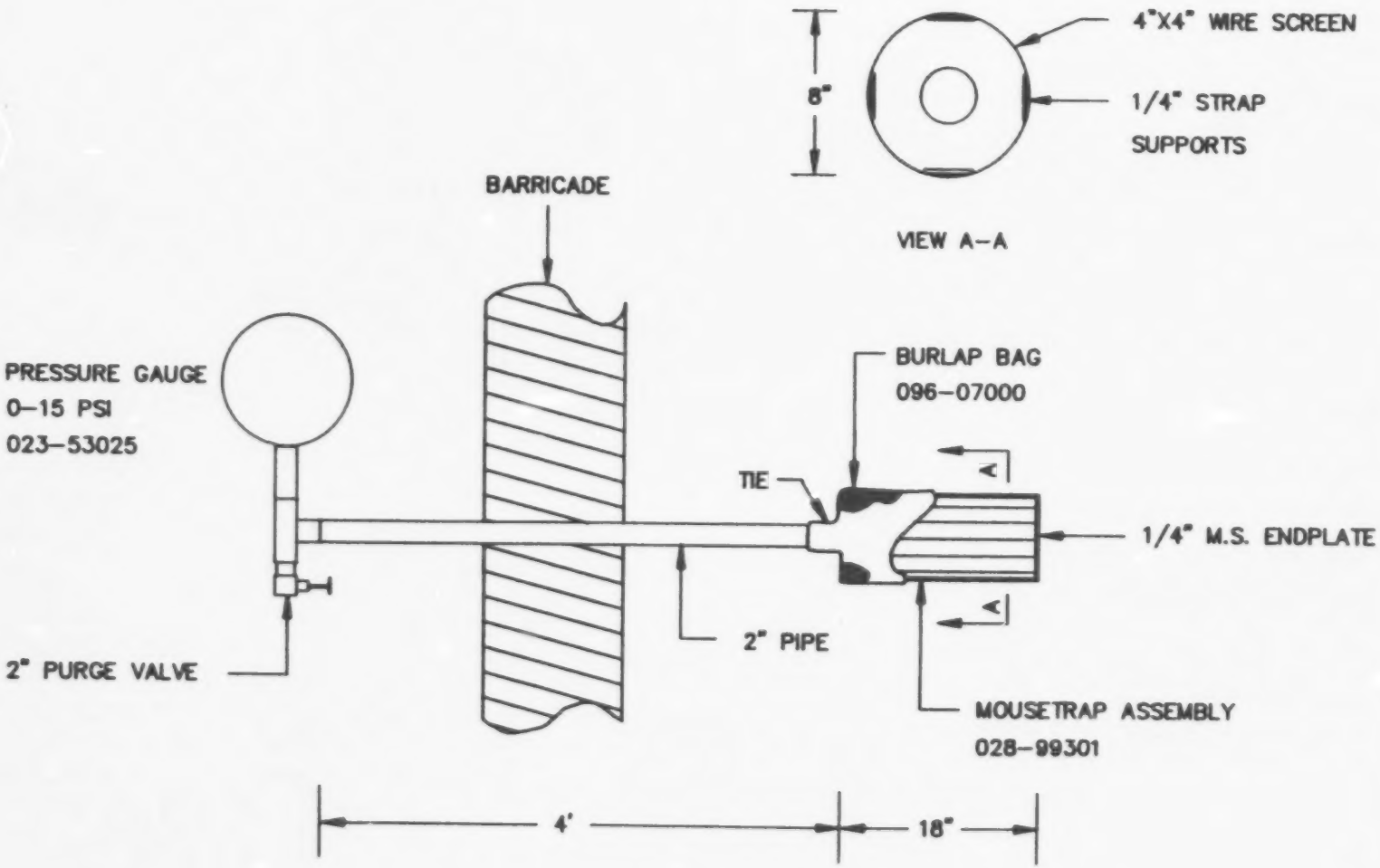
Reference: Inco, 1988

Golder Associates

VBM BARRICADE

Figure 6C-6

PROJECT NO 852-1605 DRAWN — REVIEWED DATE Nov. '89



VBM BARRICADE PRESSURE GAUGE

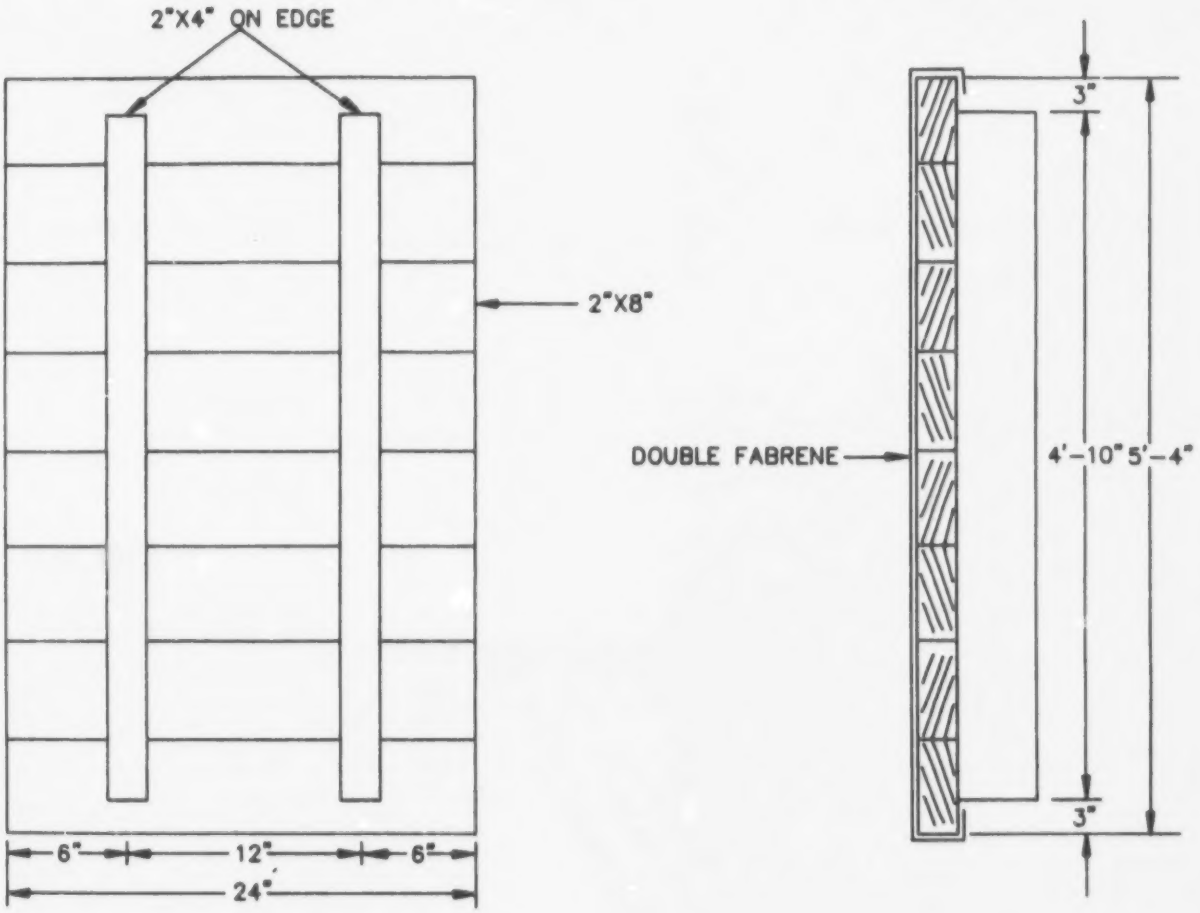
Figure 6C-7

Golder Associates

Reference: Inco, 1988

VBM BARRICADE DOOR

Figure **6C-8**



FRONT VIEW

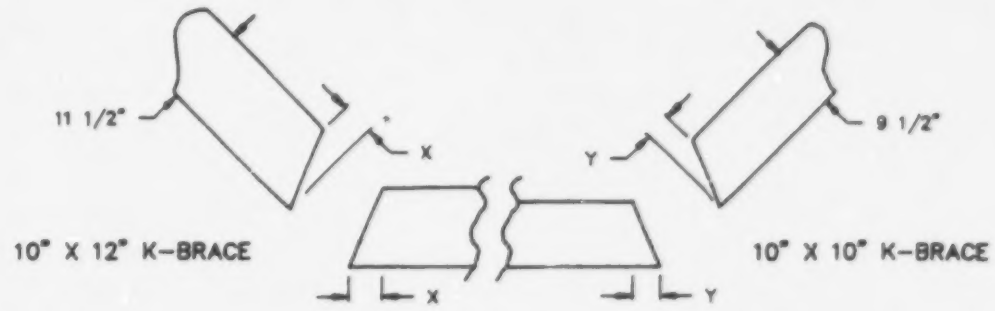
SIDE VIEW

PROJECT NO 857-1605 DRAWN — REVIEWED DATE Nov. '89

Reference : Inco Ltd.

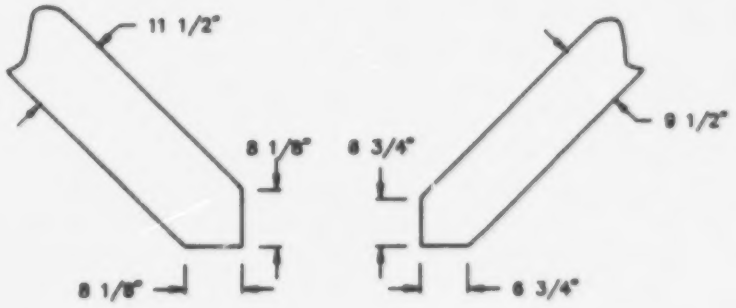
K-BRACE CONSTRUCTION CUTS

Figure **6C-9**



K-BRACE ANGLE	X LENGTH	Y LENGTH
40°	5 3/8"	4 3/8"
45°	4 3/4"	3 7/8"
50°	4 1/8"	3 1/2"

VBM BARRICADE K-BRACE DETAILS



10" X 12" K-BRACE

10" x 10" K-BRACE

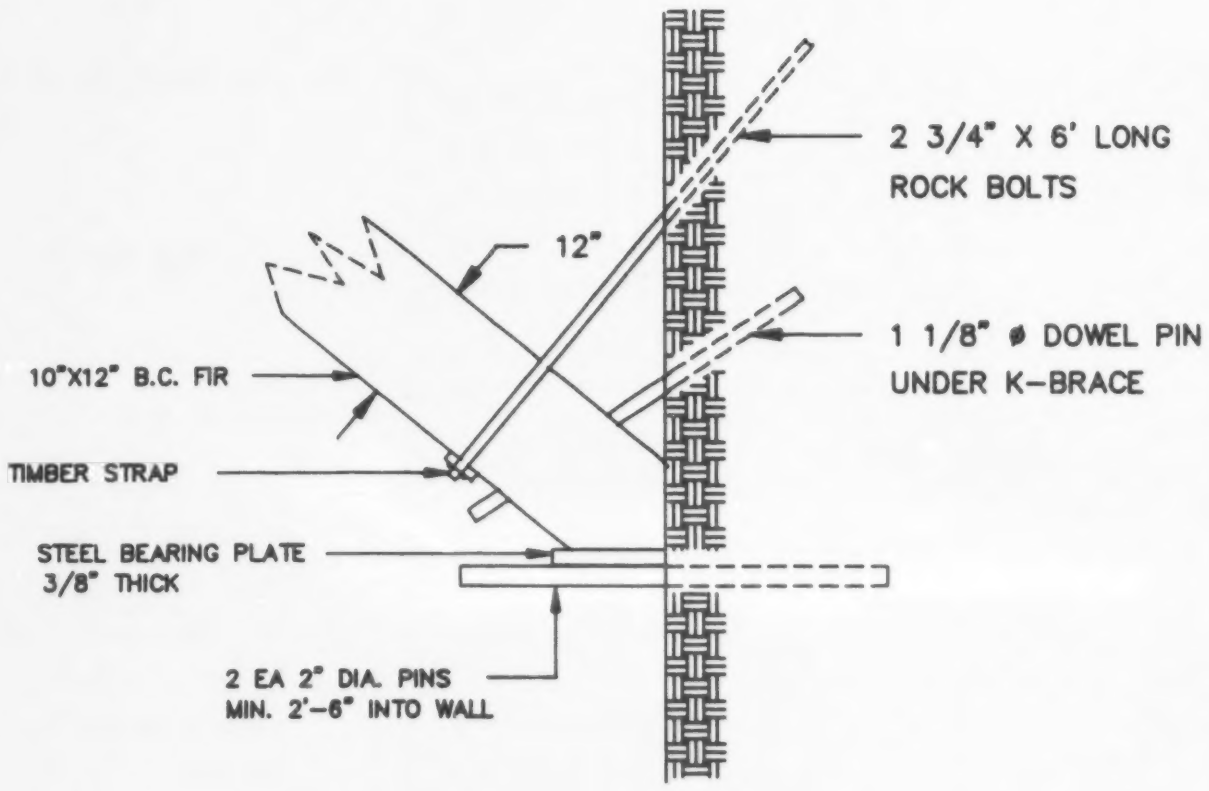
CUTS FOR PINNED END

PROJECT NO 852-1605 DRAWN — REVIEWED DATE Nov. 89

Reference : Inco, 1988

K-BRACE WITH PINNED END

Figure **6C-10**



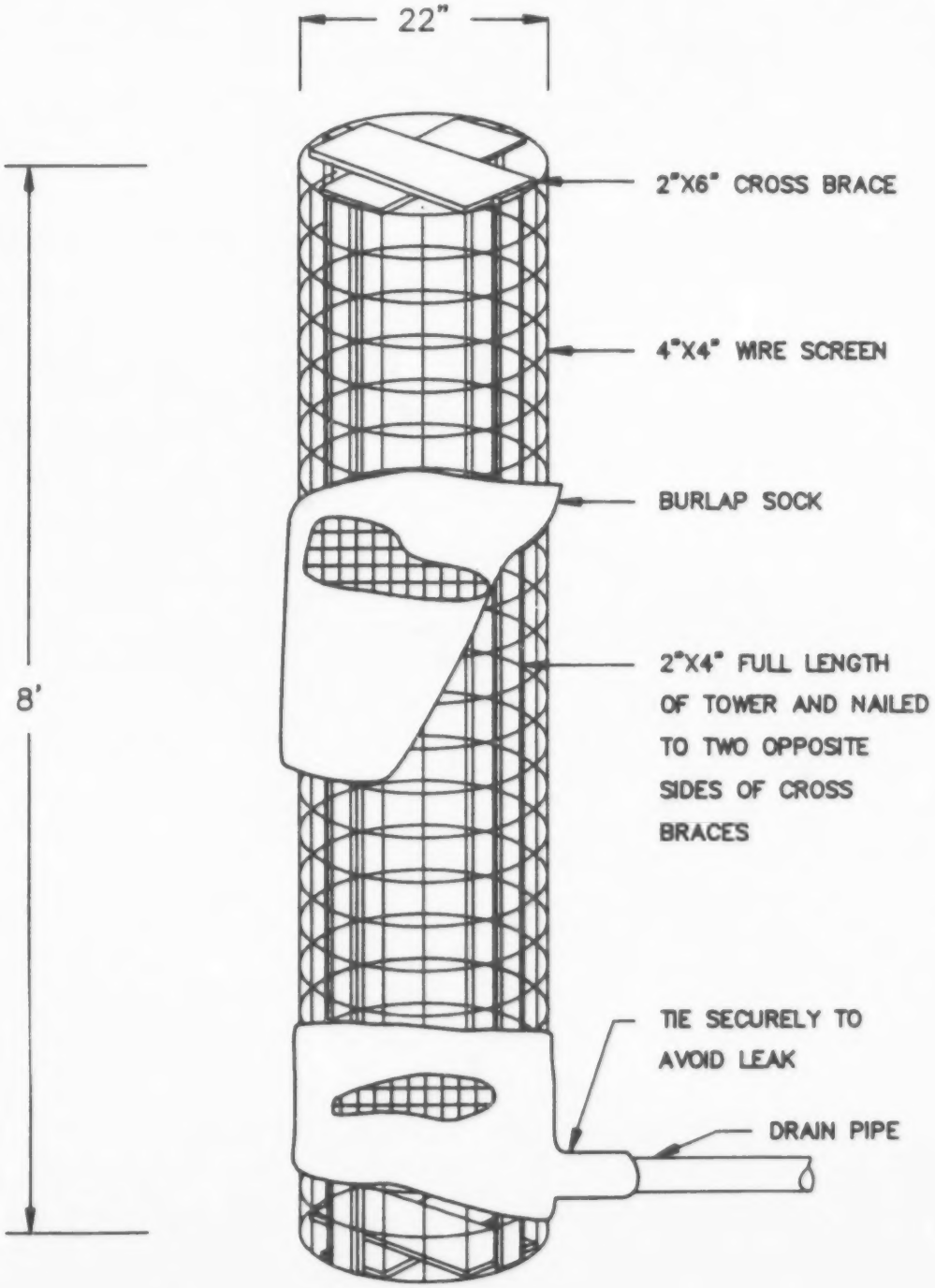
PLAN VIEW

PROJECT NO 952-1605 DRAWN — REVIEWED DATE Nov. 1989

Reference: Inco, 1988

DRAIN TOWER CONSTRUCTION

Figure **6C-11**



PROJECT NO 852-1605 DRAWN - REVIEWED DATE Nov. '89

Reference: Inco, 1988

APPENDIX 6-D

DESIGN GUIDELINES SUMMARIES FOR THOMPSON MINE, HOMESTAKE MINE, CAROLUSBERG MINE, CSA MINE

DESIGN GUIDELINES SUMMARY - THOMPSON MINE

1. Layout

Overcuts:	Drift size	=	12 ft. wide x 12 ft. high minimum
	Support	=	screened and bolted with 8 ft. mechanical bolts. Split sets and 20 - 40 ft. cable bolts depending on width and ground conditions.

Drawpoint, extraction drift, ramp and ore pass access drift dimensions vary with size of LHD used, as follows:

2 cu. yd. LHD	=	10 ft. wide x 10.5 ft. high minimum
3.5 cu. yd. LHD	=	12 ft. wide x 10.5 ft. high minimum
5 or 6 cu. yd. LHD	=	13 ft. wide x 10.5 ft. high minimum
Drawpoint support	=	rebar, with screen.
Ore drift support	=	screened and bolted with 8 ft. mechanical bolts. Shotcrete used occasionally in pour ground.
Waste drift support	=	pattern bolting with 6 ft. or 8 ft. mechanical bolts. Shotcrete in poor ground.

2. Drilling and Blasting

Hole size:	4.5 and 6.5 inch diameter
Drill pattern:	Drilling from overcut drift. Parallel hole drilling if possible, but normally inverted fan pattern with many perimeter holes non breakthrough.
Hole spacing:	5 ft. spacing, 7 ft. burden for 4.5 in. holes 8 ft. spacing, 10 ft. burden for 6.5 in. holes
Depth of charge burial:	4.0 ft. for 4.5 in. holes 6.0 ft. for 6.5 in. holes
Explosive type:	ANFO or packaged emulsion/slurry.
Charge weight:	18 lb. for 4.5 in. holes 55 lb. for 6.5 in. holes
Average advance rate:	7 ft. per blast for 4.5 in. holes 10 ft. per blast for 6.5 in. holes
Powder factor:	variable, 1.02 lb./ton ave.
Crown pillar recovery:	stop VCR when 25 ft. of ore remaining, load remaining holes with multiple charges and blast in one shot.

3. Ore Handling

Mucking:	2, 3.5, 5 and 6 cu. yd. LHD's, diesel powered
Expansion void height:	1.5 times blast volume (normally 9 feet minimum)

DESIGN GUIDELINES SUMMARY - HOMESTAKE MINE

1. Layout

Overcuts:	Drift size	=	12 ft. wide x 11 ft. high minimum
	Support	=	pattern rockbolting in narrow overcuts, 40-60 ft. cable bolts in larger overcuts
Drawpoints:	Drift size	=	10 ft. wide x 10 ft. high
	Length	=	40 ft. for LHD's
	Spacing	=	35 - 40 ft. along haulageway
Drawpoint Haulage:	Drift size	=	10 ft. wide x 10 ft. high

2. Blasting

Hole size:	6.5 inch diameter
Drill pattern:	parallel hole drilling from overcut if possible. Normally require fan drilling in deeper ore zones due to poor ground conditions preventing complete extraction of overcut.
Maximum spacing:	8 ft. x 8 ft. in competent ore 10 ft x 10 ft in weaker more fractured ore
Depth of charge burial	6.75 feet
Explosive type:	watergels and emulsions, 60 lb. charge per hole
Advance rate:	8 - 10 ft. per blast
Powder factor:	1.2 lb./ton in competent ore 0.8 lb./ton in weaker ore
Crown pillar recovery:	stop VCR when 25 - 30 ft. of ore remaining, load remaining holes with multiple decks and blast in one shot

3. Ore handling

Mucking:	3.5, 2, 1 cu. yd. LHD's (electric and diesel), Atlas Copco Cavo 320's in small block stopes
Expansion void height:	7 ft. average between back and muckpile.

DESIGN GUIDELINES SUMMARY - CAROLUSBERG MINE

1. Layout

Overcuts:	Drift size	= 4.2 m wide x 3.4 m high
	Gradient	= 1.25%
	Support	= 3 m swellex bolts and wire mesh
Drawpoints:	Drift size	= 3.4 m wide x 3.2 m high
	Gradient	= 2.5%
	Support	= 3 m swellex bolts in tight pattern and wire mesh. Concrete bullnose at drawpoint brow
Drawpoint Haulage:	Drift size	= 4.2 m wide x 3.4 m high
	Gradient	= 2.5%
	Support	= 3 m swellex bolts and wire mesh

2. Blasting

Hole size:	165 mm diameter
Drill pattern:	ring drill from overcut drift. High Stress does not permit complete extraction of overcut for parallel drilling.
Maximum spacing:	3.3 m x 3.3 m
Depth of charge burial:	2.0 - 2.5 m
Explosive type:	slurry, 1.4 gm/cc density, 24 kg charge per hole.
Advance rate:	4 m per blast.
Power factor:	0.34 kg/tonne.
Crown pillar recovery:	stop VCR when 12 m of ore remaining, develop 4 m wide slot, slash crown pillar into slot.

3. Ore Handling

Mucking:	4 cu. m LHD's
Expansion void height:	4 m between back and muckpile

DESIGN GUIDELINES SUMMARY - CSA MINE, COBAR

1. Layout

Overcuts:	Drift size	= 5 m wide x 4 m high
Drawpoints:	Drift size	= 5 m wide x 4 m high
	Length	= 15 m, driven at 60 degrees to haulage drift
	Support	= Cable dowels
Drawpoint Haulage:	Drift size	= 5 m wide x 4 m high

2. Blasting

Hole size:	140 mm for stope blastholes 165 mm for longhole raises
Drill pattern:	3 m burden x 3.5 m toe spacing See figure 6-21 for raise drill pattern
Explosive type:	ANFO in production holes Diluted ANFO in wall holes Tovex Extra in raise cut holes
Advance rate:	6 m per round in longhole raise
Powder factor:	Roughly 0.4 kg/tonne for production blasting 3.0-11.3 kg/tonne for longhole raise blasting
Crown pillar recovery:	Mass fire 5 m crown above slot. Slash longholes into slot void with column charges

3. Ore handling

Mucking:	8 yd^3 LHD's
Expansion void height:	Not specified